METAL MINING
MINING GEOLOGY

1937

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FOREWORD

Pursuing the plan of publishing at intervals TRANSACTIONS volumes dealing with a single subject or with a small group of related subjects, the present volume comprises:

1. Twenty-four papers on various phases of metal mining, including several on health and safety, a few on economic subjects, with the greatest number on mine equipment, methods, and operations.
2. Nine papers on geologic topics, most of them descriptive of structure and deposits in a single district.
3. Three papers on aviation as applied to exploration for minerals and other uses of airplanes in the mining industry.

It is believed that most of these papers will be of value to each of the large group of Institute members who are primarily interested in the search for and exploitation of metal-bearing ores.

Metal-mining papers were last published in Volume 109 (1934) and those on mining geology in Volume 115 (1935); those in the present volume have been selected from the pamphlet publications (T.P.'s) appearing in the interval. The three papers on aviation are the first sponsored by the Committee on Aviation, which was established in 1934.

The chairmen of the respective committees under whose administration most of the material accumulated and at whose instigation many of the best papers were prepared and submitted are:

Gerald Sherman, Chairman of the Committee on Mining Methods in 1935 and 1936; George M. Fowler, Chairman of the Committee on Mining Geology in 1934, 1935, 1936; Theodore Marvin, Chairman of the Committee on Aviation in 1934, 1935 and 1936; and W. E. D. Stokes, Jr., Vice-chairman of the Aviation Committee in 1936 and now Chairman.

To these chairmen and to the forty-four authors who contributed, the Institute and the profession are indebted for an excellent addition to the literature on these particular subjects.

A. B. PARSONS, *Secretary.*

NEW YORK, N. Y.
August 24, 1937

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Development, Equipment and Operation of the Blueberry Mine, Marquette Iron District

By R. S. Archibald,* Member A.I.M.E., and L. S. Chabot, Jr.†

(New York Meeting, February, 1935)

The Blueberry mine is about 10 miles west of Ishpeming, Marquette County, Michigan. The property consists of about 2000 acres in secs. 3, 4 and 5, T. 47, N.R. 28W. and secs. 32, 33 and 34, T. 48, N.R. 28W., and, excepting one intervening 80-acre tract, contains the iron formation for a length of about 2¼ miles.

The property is traversed by State Highway M-28 and is served by three railways, the Chicago & Northwestern, the Duluth, South Shore & Atlantic, and the Lake Superior & Ishpeming. The Northwestern transports the ore to a dock at Escanaba; the other railways, to Marquette. Ore for Chicago goes via Escanaba and that for Lower Lake ports, as Detroit, Cleveland, and Buffalo, via Marquette.

History

The presence of iron formation in this property has been known for many years, and mining operations on a small scale have been carried on intermittently. The Dexter mine, on the SE.¼ of the NW.¼ of sec. 3, was worked from 1883 to 1897 and 118,000 tons of ore was mined. From 1907 to 1915 extensive exploration was carried on over the whole of sec. 3, including 79 diamond-drill holes with a total footage of 26,000 ft. The Chase mine, on the NE.¼ of the NE.¼ of sec. 3, was developed from this exploration work, but after about 200,000 tons had been mined the property was abandoned. From 1908 to 1910 exploration was carried on by another company on sec. 4, 47–28, including four drill holes with a total footage of about 5000 ft. Much of the drilling on sec. 3 and of the exploration work on sec. 4 are within the present developed area of the Blueberry mine.

A general campaign of diamond-drill exploration was undertaken in 1924, and within a year a considerable amount of ore was developed. The Ford Motor Co. then took an option on the property and carried on check drilling and diamond-drill work until the spring of 1926, when enough ore had been developed to warrant the sinking of a shaft

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† Mining Engineer, Negaunee, Mich.

and equipment of the property. The Ford company took a lease and began underground development. A shaft was sunk on the NW.¼ of the NW.¼ of sec. 3 and a complete mining plant was erected. The first ore was mined in 1927. The property was closed from October, 1932 until November, 1933, at which time the North Range Mining Co. took a lease from the Ford Motor Co., began mining operations, and is now producing ore largely for the use of the Ford Company. The production of the property through the season of 1934 has been 802,863 tons.

Geology

The Blueberry mine is in the Negaunee iron formation of the Marquette Range. The district is a pitching synclinorium, which outcrops to the east near Negaunee and pitches westward. Just west of Ishpeming the iron formation in the center of the district is covered by the overlapping Goodrich conglomerate and quartzite, and it outcrops on the north and south sides of the basin from this point westward. Fig. 1 shows the outcrop in the main part of the Range. The mine is on the north limb of the basin, about 15 miles west of its outcrop.

The Negaunee iron formation has been classified as Middle Huronian. It is underlain by Siamo slate, also of Middle Huronian, and is overlain by Goodrich conglomerate and quartzite of Upper Huronian. The iron formation and other rocks of the district have been intruded by two ages of igneous rocks, the earlier probably belonging to the Huronian, the latter to the Keweenawan age. These intrusives occur as dikes, bosses and sills. Some of the igneous rocks are found only in the formations older than the Goodrich quartzite and are placed in the Huronian age; others cut through the Goodrich quartzite as well as the older rocks and are placed in the Keweenawan age. The Keweenawan intrusives appear to have an important bearing on the concentration of soft ores in the Marquette district.

The Negaunee formation consists of sideritic cherts, grunerite-magnetite schists and ferruginous chert. Little slaty material is found within it. A zone of from 50 to 100 ft. on top of the Siamo slate is somewhat slaty and is known as transition formation; but all the rest of the material is cherty. It is supposed that this formation originally consisted largely of sideritic cherts, which, in the vicinity of intrusives, were altered by heat and pressure into grunerite-magnetite schists. Secondary alteration by meteoric waters has changed the bulk of the Negaunee formation into ferruginous cherts.

From a commercial standpoint the iron formation is divided into hard-ore jasper and soft-ore jasper, the former being the upper part of the formation, lying directly under the Goodrich conglomerate, and the latter being the lower part. The iron formation was laid down horizontally as a sedimentary rock, and secondary concentration began in this period.

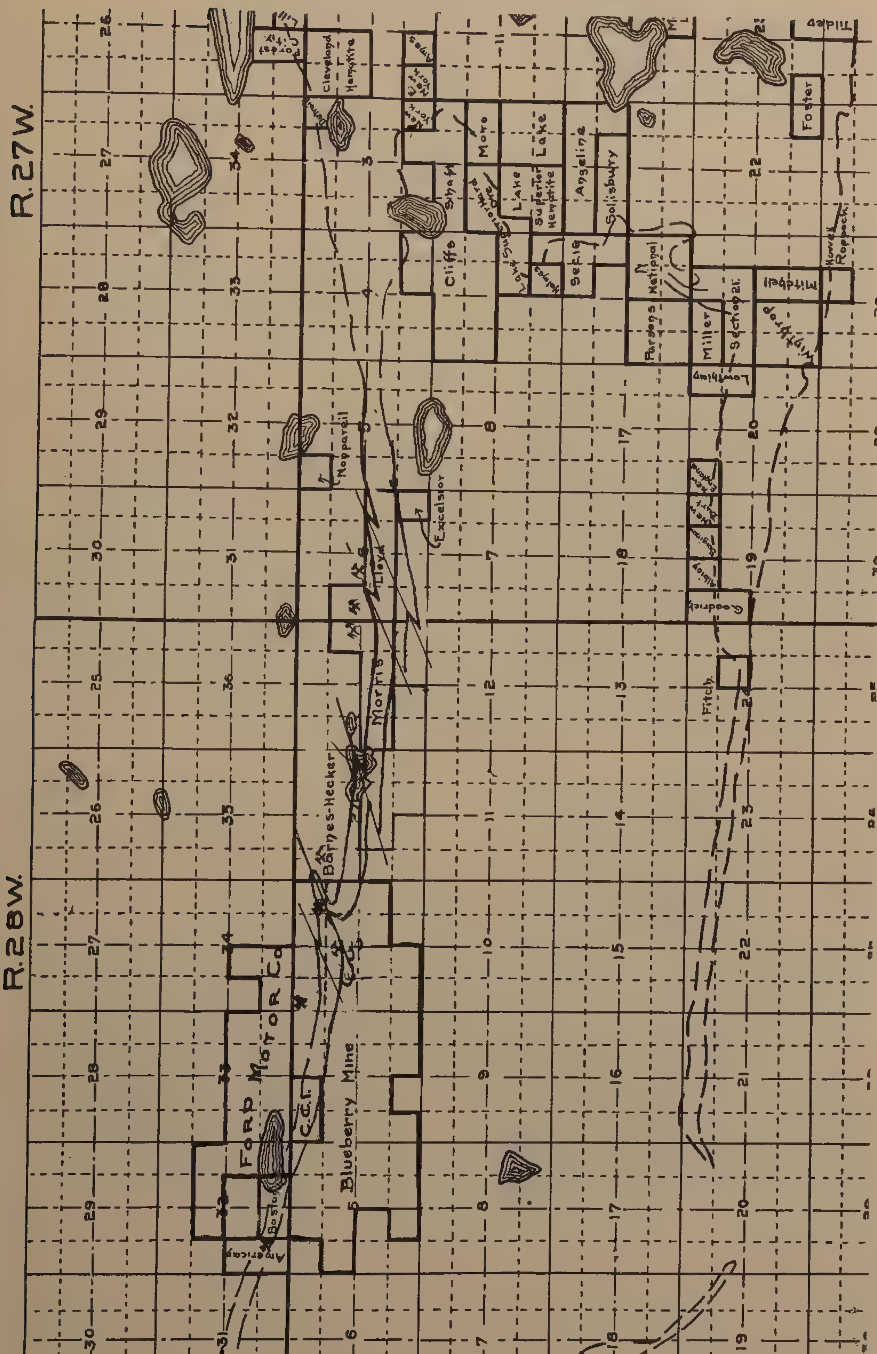

FIG. 1.—PART OF THE MARQUETTE IRON-BEARING DISTRICT.

The upper portion was oxidized and some concentration occurred while the rocks were only slightly folded. Later, extensive folding took place. By this folding and by igneous intrusions, the oxidized iron formation and the ores already deposited in the upper part of the formation were altered and hardened into hard-ore jasper, hard hematites and magnetites.

After the folding and the faulting that accompanied it had taken place, oxidation and secondary concentration began in the lower part of the formation; where the circulation of meteoric waters was made possible by the breaking up of the rocks, the formation was completely oxidized into a ferruginous chert, and where structural conditions were favorable commercial bodies of soft ore were formed by the leaching out of the silica and the deposition of iron oxide.

According to early theories, the Marquette ores were deposited in troughs due to minor folding in the Siamo slate footwall, or in troughs formed by the intersection of dikes with the slate footwall. These structural features are favorable to the concentration of iron ore, but it is now believed that the breaking up of the formation by close folding and faulting has been the principal cause of secondary concentration. In many places on the Range the orebodies are not formed directly in the troughs but occur in vertical zones, where close folding or faulting has shattered the rocks so as to permit the ready circulation of meteoric waters.

The strike of the North limb of the Marquette synclinorium, on which the Blueberry mine lies, is slightly North of West for a distance of 6 miles West of Ishpeming, and the prevailing dip is almost vertical. Along this limb the width of the iron formation is from 500 to 1000 ft. Cutting across it is a series of oblique faults, which strike North of East and dip steeply to the northwest. These faults, ranging in throw from a few feet to 2000 ft., displace the formation on the West side to the North. One fault in the eastern part of the Blueberry property, in the location of the old Dexter mine, has a throw of about 2000 ft. Some of the faults follow along planes occupied by the diabase dikes; others cut across the dikes and displace them. The fault planes are the loci of the orebodies along this rim of the basin, and along them are situated the orebodies in the neighboring East Lloyd, Norris, Chase, Dexter and other mines. The cross-sections on Figs. 4, 5 and 6 show that the Blueberry orebody does not lie in the trough formed by the intersection of the dike with the footwall, but in a brecciated zone between the slate footwall and the dike. The ore is often directly on the dike; rarely on the footwall, which is vertical, while the dike dips about 75° to the northwest. The longitudinal section (Fig. 7) shows that at three points the orebody rises considerably above its average height; reaching practically to the surface at the easterly end. Just east of this is the outcrop of the intersection of the dike and footwall. At other points, where the orebody rises above its average height, small transverse faults intersect the formation. The westerly

pitch of all orebodies along the north rim of the basin results from the northwest dip of the fault dikes. Fig. 8 shows the relations described, on the 700-ft. level, between the footwall, dike and transverse faults.

Surface Layout and Equipment

The surface plant was designed by experienced mining engineers of the Lake Superior district, in cooperation with factory engineers of the

Fig. 2.—Looking north, Blueberry mine.

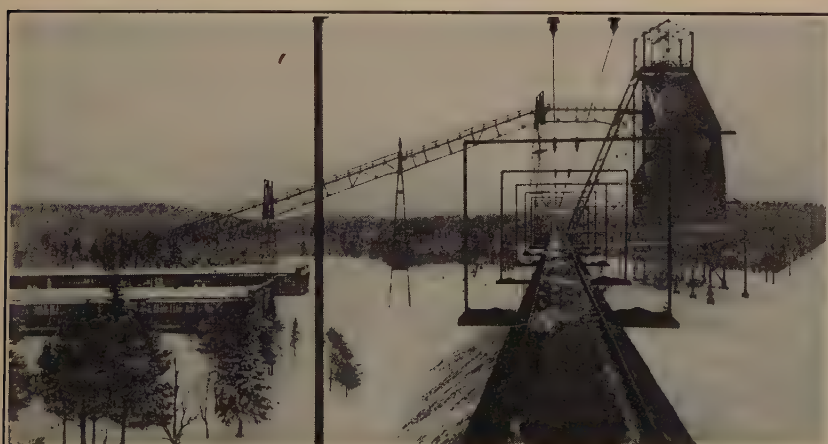

Fig. 3.—Looking east, Blueberry mine.

Ford Motor Co. It exhibits the best features of local practice, together with many ideas derived from the Ford factory practice. The result is a plant somewhat different from others in this district. Fig. 2 is a general view of the building and the ropeway leading from it to the headframe; Fig. 3 shows the permanent steel stocking trestle and the head-

frame. In the Lake Superior district several separate buildings usually house the office, shops and dry house. The Blueberry layout includes all these in one large U-shaped building, the base of the U being 210 ft., and the width 40 ft. Two wings are 180 ft. long and 50 ft. wide. Between them the ropeway leads from the engine house, in the base of the U, to the headframe, which is about 800 ft. distant. The flat-roofed building

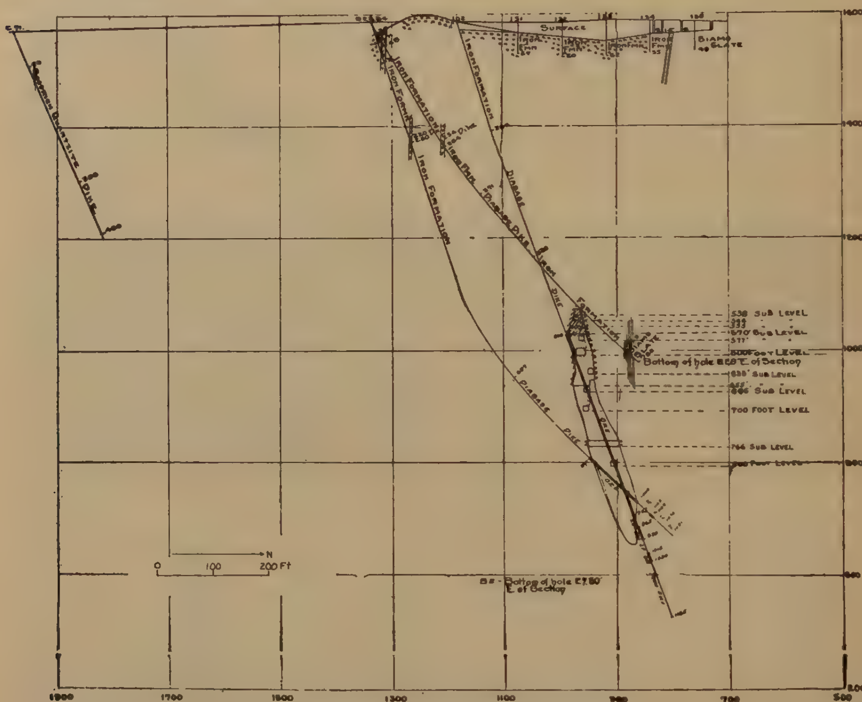

FIG. 4.—CROSS SECTION 4368 FEET WEST.

is 16 ft. high, of red brick, steel and glass. The upper two-thirds of the wings are glassed in with large windows, like those in factories, which provide excellent lighting. A disadvantage of this design is that in a cold climate like that of the Lake Superior district it is more expensive to heat than the usual type of construction.

The east wing contains the office, laboratory, first-aid room, foremen's change room, underground dry, washroom, and surfacemen's dry. In the base of the U are the heating plant, hoisting engines, air compressors, motor-generator set for direct current, and switchboards; the west wing contains the shops, warehouse, storeroom, garage and sample-grinding room.

The equipment in the office, laboratory, first-aid and change rooms is the standard equipment used in the Lake Superior district. The only

noteworthy feature is the charging room, located in the underground change room, which contains charging racks to accommodate 125 Edison storage-battery electric lamps, used underground instead of the carbide

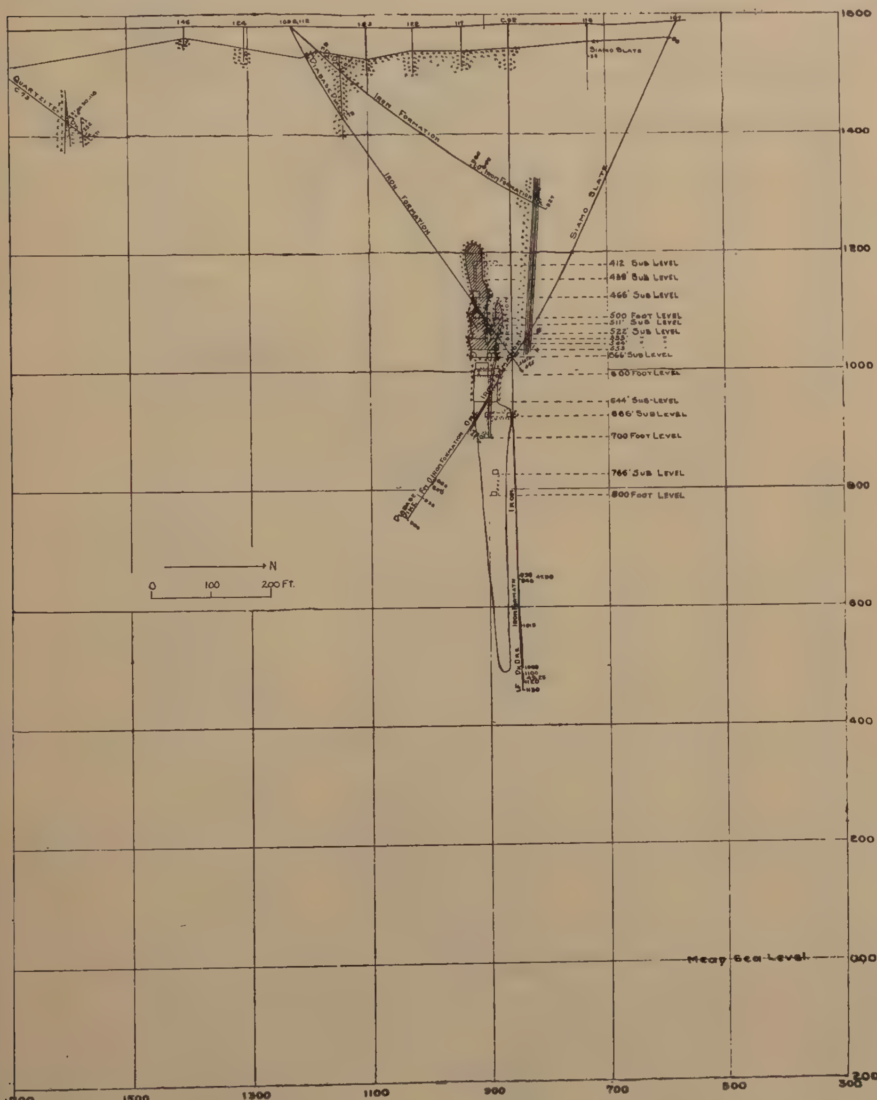

FIG. 5.—CROSS SECTION 4966 FEET WEST.

lamps usually used in the district. The electric lamps give a brighter light and are not extinguished by a premature explosion of powder or an air blast underground, thus providing light at all times for the escape of miners from dangerous conditions. The heating plant consists of two locomotive-type boilers of 100 hp. each, with sufficient capacity to take

care of underground pumping in case of failure of electric power. As electric power has been found sufficiently trustworthy to make steam

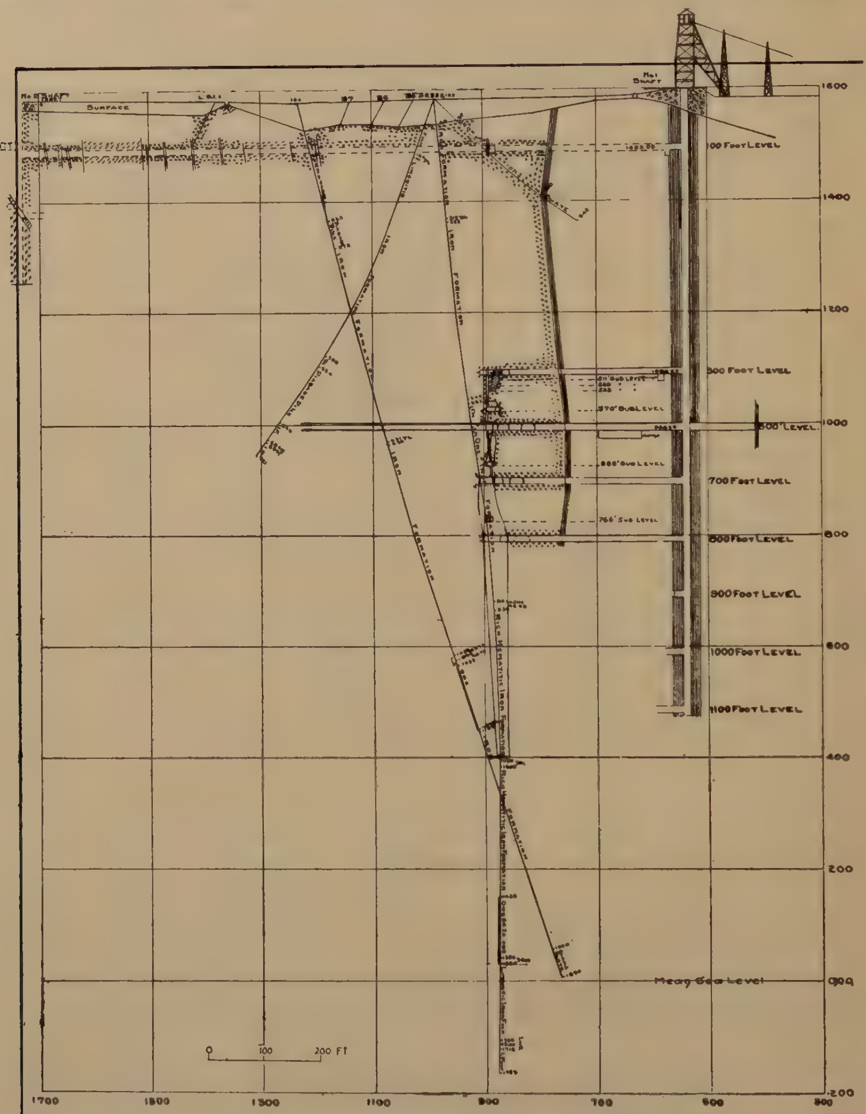

FIG. 6.—CROSS SECTION 5200 FEET WEST.

pumps unnecessary, the boilers are used entirely for heating. Only one boiler is operated at a time, furnishing steam at 15 lb. pressure.

The engine house contains the following equipment: (1) One Chicago pneumatic, 2200-cu. ft. air compressor; (2) one Worthington, 1058-cu. ft. air compressor; (3) one cage hoist, Lake Shore Engine Works, single

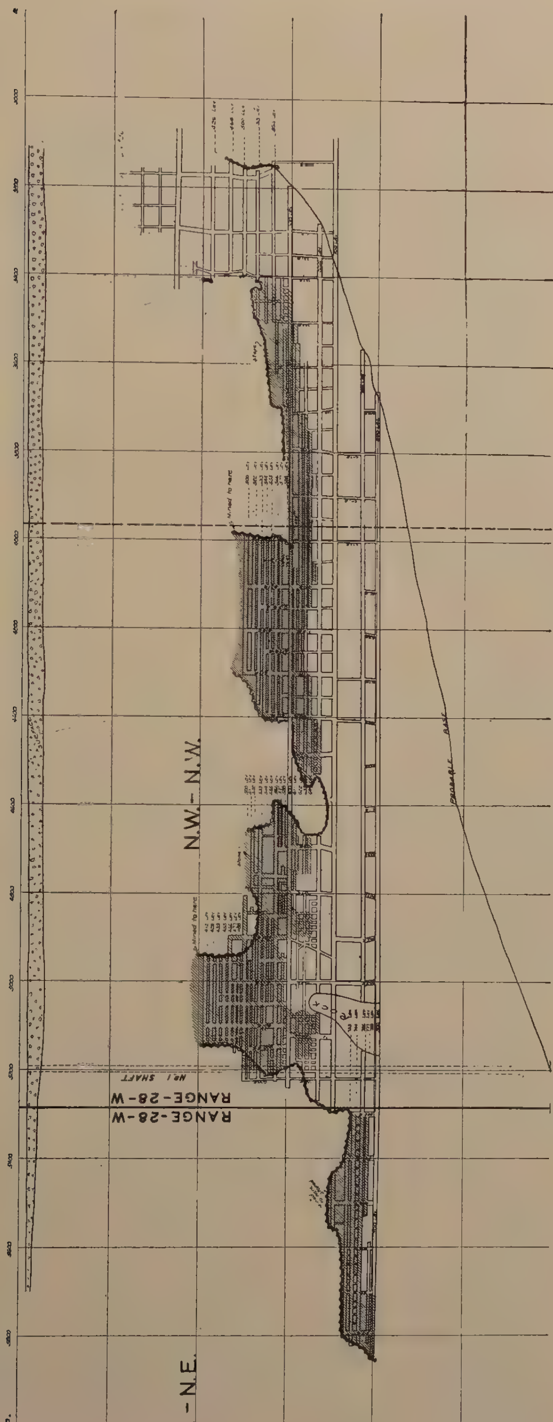

FIG. 7.—LONGITUDINAL SECTION, BLUEBERRY MINE.

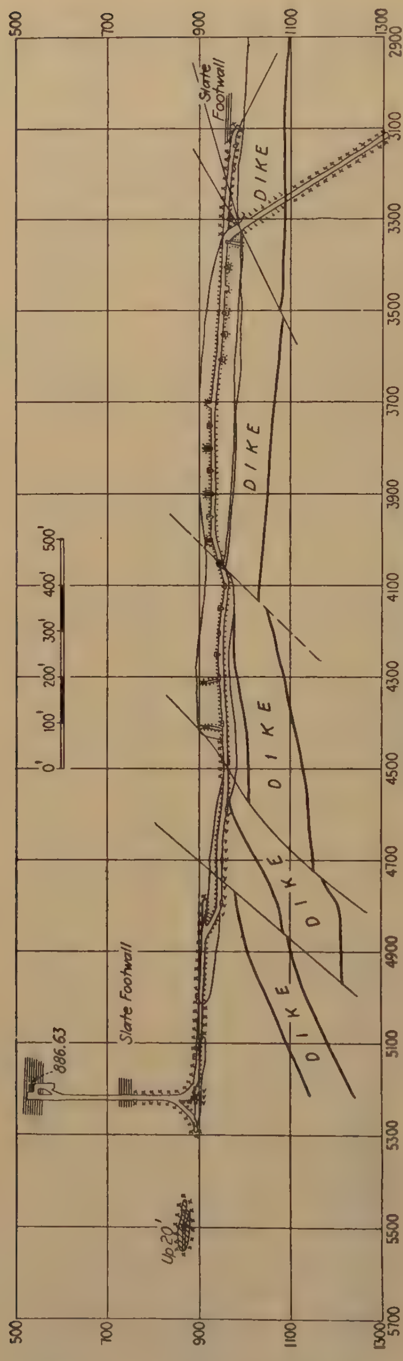

FIG. 8.—PLAN OF 700-FOOT LEVEL.

drum, 10 by 10 ft., rope capacity of 1600 ft. of 1¼-in. wire rope, equipped with Lilly control and with cage operated in balance with a dummy; (4) one skip hoist, Lake Shore Engine Works, single drum, 10 by 10 ft., capacity of 1600 ft. of 1¼-in. rope, with Lilly control and operating two 4¼-ton skips in balance; (5) one 200-kw. motor-generator set, for underground lighting, haulage, and scraping. There are the usual switchboards for this equipment, and a switchboard for operating an automatic fire pump on the first level, which furnishes the water supply for the building and for drinking purposes. It pumps into a 60,000-gal. elevated steel tank.

A narrow-gage track from the cage compartment of the shaft passes through the center of the blacksmith shop, and back through the machine shop, carpenter shop and warehouse situated in the west wing. At the rear of the building a standard-gage track delivers heavy equipment to the mine, and passes on to the rear of the engine room, where coal is delivered to the boiler house.

The blacksmith shop contains an Ingersoll-Rand No. 50 drill sharpener, one oil-fired forge, two coke forges, a punch and shears, a 20-in. shop grinder, a 48-in. grindstone, a steel tempering tank, and a 600-lb. steam hammer.

The machine shop, which is not partitioned off from the blacksmith shop, contains a 27-in. lathe, a 12-in. lathe, a power hacksaw, a radial drill press, a pipe and bolt threader, a pipe

threader (2 to 14-in. pipe), a 12-in. shop grinder, four cast-iron work benches, a 100-ton wheel press, and an overhead crane over the pipe machine.

Behind the machine shop and in the same room is the carpenter shop, with rip saw, band saw, planer and metal shelving for tool boxes. All engines in the shops are operated by individual motor drives, with push-button starters. Shop motors are of 115 volts. All wiring is in conduits and switches are placed 9 ft. above the floor. Oxyacetylene welding is used in the shops and underground. Beyond the carpenter shop and enclosed by wire screening is the warehouse, with a small room enclosed for greater warmth as an office for the warehouse man. The rest of the warehouse has steel shelving. Along the narrow-gage track in the warehouse is space for storing heavy equipment, as motors, underground hoists, pumps, etc. In the northwest corner of the building is a garage, with stalls for five motor units. The oil supply for the plant is also housed in the garage in steel barrels.

The sample-grinding room is in the extreme northwest corner. It contains a steam drying oven, small gyratory crusher, pulverizer, automatic sampler and steel benches. As the grinding of iron-ore samples is very dirty work, this room is in the least conspicuous part of the plant.

The timber yard is south and west of the shaft house, on a terraced area of about 600 by 75 ft. A high-line standard-gage track runs on the south side of the timber field, and all timber can be unloaded by gravity. A narrow-gage track on the north side of the timber field, about 3 ft. below its floor, leads to the cage compartment of the shaft. After being framed, timber can be rolled directly on to mine trucks without lifting. A snow shed of galvanized steel protects this timber track. The south side of the shed is open for the loading of timber trucks. Fire hydrants with coils of hose attached are placed at regular intervals along the north side of the timber field.

The mine timber used at the Blueberry mine is as follows: For all slicing operations, 8 to 10-in. green hardwood 9 ft. long is used. Props are cut 8 ft. long. Main-level drift sets are of green hardwood or tamarack, 10 to 12-in. tops and 9 ft. long. Covering poles for slices are of green tamarack, 3 to 4-in. tops and 9½ ft. long. Cribbing timber is green tamarack, 5 to 7-in. tops, and cut in lengths of 5 ft. 4 in., 10 ft. 8 in. and 16 ft. Seven-foot round spruce and tamarack and split cedar lagging is also used.

Headframe.—The headframe is a four-post steel structure, with the usual back braces. Its height, from shaft collar to center of hoisting sheaves, is 124 ft. 8 in. It is fully covered with painted corrugated sheet-iron siding. A Telsmith No. 16A gyratory crusher, mounted on timbers bolted to the headframe, is set to crush to 3 in. It is driven by a 75-hp. General Electric induction motor, with a 7-rope Texrope drive. The

headframe contains two steel pockets, for loading into railway cars on either side of the shaft house. Landing floors, from which ore is transported to the steel stocking trestle, are built on both sides of the skipway, so that ore or rock can be loaded from either side for stockpiling.

Five idler stands carry the ropes from the headframe to the engine house. The first of these is on top of the back braces of the headframe; the others are independent idler stands, and all are connected to the headframe by a catwalk, which permits the attendant to go from one stand to another to grease and adjust the sheaves.

Stockpile Trestle.—The stockpile trestle is of steel, of a single-leg type. It is 42 ft. 6 in. high from the stockpile floor to the rail. There are ten 80-ft. spans, giving a total length of 800 ft. The trestle has a single track and a capacity of 125,000 tons. On the stockpile floor two railroad tracks, one on each side, serve for stockpile loading, and are shifted as the loading progresses.

A larry car of 90-cu. ft. capacity, operating by remote control from the shaft house and having magnetic brakes, takes ore from the skip pockets and delivers it to the stockpile ground. Dumping blocks can be placed at any point on the trestle for dumping the larry car. A second trestle of wooden bents runs parallel to the steel trestle, for additional ore storage. It is of the same height as the steel trestle and accommodates the larry car mentioned.

Electric Shovel.—The stockpiled ore is loaded out in the summer months with a caterpillar-mounted Marion electric shovel (model 490), which has a 2-cu. yd. dipper. This shovel loads into standard railway cars at the rate of 10 cars per hour, and is novel equipment for stockpile loading at iron mines, where steam railway-type shovels are customary. The latter require seven men for loading; the electric shovel requires only three men.

Electric Substation.—The mine is completely electrified, the only coal used being for heating purposes. Electric power at 33,000 volts is purchased from the Cliffs Power and Light Co. It is stepped down to 440 volts by a substation at the east side of the building, which contains three 500-kva. Maloney transformers, master switches, oil circuit-breakers, and lighting arresters, and is enclosed by Cyclone wire fencing.

Shaft.—The mine has one five-compartment shaft, sunk in the footwall about 350 ft. north of the orebody. It is 15 ft. by 10 ft. 10 in. inside dimensions, and contains two skip roads, parallel to the width of the shaft, 4 ft. 3 in. by 5 ft. 1 in. in dimension and a cage compartment 10 ft. 10 in. by 6 ft. There is also a dummy compartment and pipe and ladderway. The shaft is lined with steel sets of 6-in. H-section, spaced 5 ft. apart vertically, and lathed with 3-in. fir plank. Bearing sets of 10-in. I-section are placed at 100-ft. intervals.

The cage is of steel, covered with wire screening and having sliding steel-plate doors. It accommodates 30 men, and will also take trucks loaded with the maximum length of timber used in the mine. The skips are of Kimberley type, of 90-cu. ft. capacity, or about 4½ tons.

The shaft is 1100 ft. deep. A drainage level, from the main shaft to a small auxiliary shaft about 1200 ft. south, has been driven on the 100-ft. level. By holes drilled to the drainage level, considerable water is caught at this elevation and pumped to surface. At this point is also located the automatic water-supply pump described earlier in the paper. Operating levels are driven at 500, 600, 700 and 800 feet.

Loading Pockets.—Loading pockets originally constructed on the 600, 700 and 800-ft. levels were of one-car capacity and considerable delay in dumping a trainload of ore was caused by waiting for the skip. A new pocket now being constructed on the 800-ft. level, with a capacity of 29 cars, consists of a main pocket divided into two parts, below which are measuring pockets for loading each of the skips; it is of steel and the main part of it can be moved to a lower level after mining operations have been completed above the 800-ft. level.

Pumproom and Equipment.—The pumproom and sump are on the 800-ft. level. There are four three-stage centrifugal pumps, capacity 500 gal. at 500-ft. head, each driven by a 100-hp. Westinghouse motor. Two pumps are connected in tandem and make up a single unit, with a capacity of 500 gal. and 1000 ft. head. Water is delivered to a 10-in. discharge line, leading to a wooden launder carrying the water ½ mile to the south, where it is run into a small lake. The present pumping equipment is not satisfactory for dirty water, and it is proposed to install on the tenth level an electric reciprocating plunger pump, with a capacity of 1000 gal. against a 1000-ft. head.

The sump on the 800-ft. level from which these pumps take their water is 70 ft. long, with a cross-section of 14 by 16 ft. As its storage capacity is small, an unused drift on the west side of the 800-ft. level has been dammed off for additional storage. All pumping is done between shifts, so that the pumps are not operated on the peak load. The average amount of water raised is 450 gal. per minute.

Locomotives and Cars.—Three 6-ton Goodman underground trolley locomotives are in use, each hauling a train of eight 76-cu. ft. rocker dump cars. A small Mancha battery locomotive is used for handling timber, tools and other supplies. The underground tracks are of 40-lb. rail with 30-in. gage.

Tugger Hoists and Scrapers.—Each mining place has a tugger hoist and 42-in. scraper, for moving ore to the raise. The hoists now in use are 7½-hp. Sullivan tuggers, 10-hp. Sullivan tuggers and 15-hp. Ingersoll-Rand tuggers. The smaller Sullivan hoists operate 30-in. scrapers in

small development drifts; the larger hoists are used exclusively for slicing. The standard scraper is of hoe-type construction, 42 in. wide, of manganese steel, and without teeth on the lip, which is reversible.

Drilling Equipment.—Drilling in ore is done by Ingersoll-Rand RB-12 augur machines, using diamond section augur steel. Raising in hard ore and rock is done with a Jackhamer, mounted on an air-feed stoper leg and using ⅞-in. hexagon hollow steel. Drifting in rock is done by Ingersoll-Rand, column-mounted, N-72 drifting machines with 1¼-in. hollow round steel.

Mining Methods

Level Development.—The average width of the orebody is about 50 ft., its developed length on the seventh level being about 2000 ft. Fig. 8 is a plan of the seventh level development. The main haulage drifts are at 100-ft. intervals, driven directly in the orebody. All drifts in ore must be timbered and the uniform drift sets have 9-ft. caps and 9-ft. legs, the latter set with a batter of about one inch per foot. The sides and backs of the drifts are lagged with poles or split cedar lagging. Raises are driven up from the main-level drift at intervals of 100 ft. and are inclined at 65°. They have two compartments, each 4 ft. 4 in. square inside of timber. All raises are cribbed. The length of the raise is along the strike of the orebody. Steel disks operated by air cylinders are used in the ore chutes.

Sublevel Stoping.—In the top of the orebody, or where the width of the orebody is not sufficient for economical top-slicing operations, sublevel stoping is employed. A scraping level is established at the lower limit of the stope and a standard timbered drift is driven longitudinally through the middle of the orebody. Small dog raises are driven in the orebody at intervals of 100 ft. Mills at 15-ft. centers are put up to a height of 20 ft. above the scraping sub on either side of the drift. Dog-drift sublevels are driven connecting the tops of these mills and also at 20-ft. intervals above. These sublevels are driven between the 100-ft. raises. Stoping is then begun at one end of the block of ore, the first raise being the start of the stope. The stope is carried back evenly on each sublevel. The men in the stope are protected by the brow of the bench above them, and the 20-ft. intervals between sublevels permit the drilling of upper and lower holes to break all the intervening ground. The maximum height of stopes above the scraping sublevel is 200 ft. The subdrifts are about 5 ft. high and 4 ft. wide, and in them are used 7½-hp. slushers with 30-in. scrapers. This method of mining is used where the orebody is under 50 ft. wide, and unless there is a dike along the hanging wall or passing through the stope the recovery and the grade of the material produced is very good.

FIG. 9.—SUBLEVELS ABOVE THE SEVENTH.

Top Slicing.—Radial top slicing is the principal method of mining. After sublevel stopes are completed in the top of the orebody, the scraping sublevel becomes the first sublevel for top-slicing operations. Pillars between the floor of this sublevel and the floor of the stope are removed by slicing and the floor of the sublevel is covered down. Thereafter, slices are spaced at vertical intervals of 11 ft. The first slice is started from the raise by crosscutting north or south to the wall, with succeeding slices side by side, each driven to the rock boundary or to the limit of the block. Each gang or contract has a block 50 ft. east and 50 ft. west of the raise, the block having an average width of about 50 ft., making a total block for each contract 100 ft. long and 50 ft. wide. Fig. 9 shows the method of radial slicing. In the upper sublevel is shown a rectangular system of top slicing, which was used originally in the mine, but as this involves double handling of material in scraping it was abandoned in favor of radial slicing.

Each slice is timbered with sets at 5-ft. centers. These are the standard timber sets, 9-ft. caps and legs. The timber is green hardwood. Poles are placed on top of the timber and on these are laid ⅝-in. hardwood boards, to prevent small chunks from dropping on the miners. In advancing slices in loose ground forepoling is used. After a slice has reached its limit it is covered down by 9½-ft. tamarack poles, which are laid on three crosspieces and spiked. The poles overlap and are spaced the width of a pole apart. After they have been laid wire netting is nailed over them. The netting covers both the floor and the solid side of the slice. It ties the matting together and prevents rock from running into a new slice from the one just mined. After a slice has been covered down the timber legs are drilled and blasted out before the next slice is driven.

Drilling and Blasting.—The character of the ore varies in different parts of the mine, but as a whole it is hard ground for augur drilling. In solid drifts, from 20 to 25 holes are required to break a 10 by 10-ft. section; in slices with one open side, 15 to 20 holes are needed. The holes are drilled about 6 ft. deep and break 5 ft. of ground. Drilling speeds range from 0.45 to 1.09 ft. per minute. The average drilling time for a round is 4½ hr.; in some parts of the mine, as much as 6 hr. Experimental work is being carried on to increase if possible the drilling speed, so that a complete cycle of drilling, blasting, mucking and timbering can be finished in an 8-hr. shift. To accomplish this, the maximum drilling time must be 4 hr. The miners, two men in each contract, carry on all of the operations in the working place. They hoist their timber with the slusher hoist from the tramming level, drill, blast, muck and timber. Clay tamping cartridges, of the same shape and wrapping as the dynamite, are used for stemming.

Contract System.—All miners, both on development and mining, are on a straight contract system. Payment for development work, which consists largely in driving raises and small sublevel drifts, is on a footage basis; large drifts and slices are on a car basis. The miners are charged for their explosives but not for other supplies.

Loading, Tramming, Hoisting and Stockpiling.—The ore from the breast in the top-slicing contracts and the ore from the mills in the scraping subs is transported to the raises by slusher hoists and 42-in. scrapers. The wire ropes for these scrapers have a metal core; the pulling rope is ½ in., the tail rope, ⅜ in. Their average life is 1½ months. The ore from the raises is loaded into 76-cu. ft. cars, trammed to the shaft in eight-car trains, hoisted by two 4½-ton Kimberley skips in balance, and dumped in the headframe on to a 3-in. grizzly. Oversize from the grizzly goes through a No. 16A Telsmith crusher and joins the undersize either in the pockets, which are used for loading railway cars, or in the small pockets from which it is dumped into the larry car. In stockpiling, ore is transported by a 90-cu. ft. larry car to the stockpile trestle. This larry is operated from the shaft house. Because of the presence of free moisture, ore is stockpiled so far as possible prior to shipment. The ore loses from 1 to 2 per cent of its moisture in the stockpile, and the resulting improvement in grade more than compensates for the small cost of stockpile loading.

Product

Only one grade of ore is produced, known in the trade as an Old Range Non-Bessemer grade, and having the following approximate analysis in its natural condition: Fe, 50.00 per cent; P, 0.086; Mn, 0.60; Si, 7.50; Al_2O_3, 2.86; CaO, 1.21; Mg, 1.17; S, 0.010; moisture, 10.00; loss by ignition, 5.64.

Chemically this is a particularly desirable non-bessemer grade, because of the ratio of silica to alumina. It is known to furnace men as a well balanced ore, and smelts readily on account of the high loss by ignition. The combined water is driven off in the upper part of the blast furnace, leaving the ore in a porous condition, whence it is readily reduced by the furnace gases.

Summary

The equipment, development and operation of the Blueberry mine has been carried on along lines quite similar to other mines in the Lake Superior district. The surface layout, however, is more extensive and elaborate than that at most mines, and can be justified only by the expectation of a long life for the property. The present developed

reserves are substantial, and geological conditions point to a continuation of the orebody to great depth.

The most striking feature of the surface plant is the substantial and attractive design of the building. The grounds surrounding it have been extensively landscaped; the railway embankments have been sodded and in summer the building is surrounded by flower beds. Above all, the plant is economical in operation.

Slot System of Mining at Golden Queen Mine, Mojave, California

By Charles A. Kumke*

(New York Meeting, February, 1937)

The "slot" system of mining in use at the Golden Queen mine, Mojave, Calif., does not involve any new mining methods. It is, however, a new combination and adaptation of several stoping systems in common practice.

The first plans for stoping operations at the Golden Queen mine called for ordinary shrinkage stopes. The hanging wall of the vein, where exposed, is a very hard felsite with a dip varying from 50° to 70°. The vein itself is quartz, shattered and fractured enough to make it easy to drill and blast to a clean hanging wall. There is no well defined footwall. The footwall limits had to be determined by assays as the metal content faded out into the felsite.

Two shrinkage stopes, each approximately 200 ft. long, were laid out and started on the 200-ft. level. One of these stopes, varying in width from 8 to 18 ft., was carried on up to the top limits of the ore, approximately 120 ft., without a break in the operation. The shrinkage system proved entirely satisfactory and low costs resulted from this operation. In the final drawing of this stope, two timbermen followed down on the ore, scaling the hanging wall clean of small fragments and placing an occasional long stull from foot to hanging wall where it was deemed necessary. Few such stulls, however, were required. The hanging wall stood up very well until the stope was entirely emptied of ore. Waste fill was then put into the stope by the glory-hole method, through a waste raise that had previously been run to surface from the middle of the stope.

The second shrinkage stope ran into difficulties about 35 ft. above the sill floor. Up to this point the maximum width of the ore had been about 15 ft. The ore suddenly widened into the footwall to a distance of 40 ft. in the north end of the stope. In extending the stoping operations over this width, it was soon found that the vein material would not arch and support itself. Drilling in this end of the stope was then entirely too hazardous, and all operations were stopped. The ore was drawn through the chutes below as rapidly as possible and as soon as the stope was entirely

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* Mine Superintendent, Golden Queen Mine, Mojave, Calif.

emptied of broken ore waste fill was put into the stope through a raise previously run through to surface. Here also the waste was mined by the glory-hole method. To get a level spread of the waste in the stope, slusher hoists and scrapers were used. A number of temporary timber cribs were hurriedly run up from the top of the waste fill to the back of the stope to prevent further sloughing. These cribs were then replaced by regular square-set timbering.

This trouble area involved the north end of the stope for a distance of about 65 ft. and, as this end of the stope had been a little ahead of the remainder of the stope when the widening of the ore into the footwall started, it was foreseen that the same conditions might be encountered in the rest of the stope a little higher up. If then the back of the stope would not arch and support itself in a long stope when the ore was over 18 ft. wide, it was out of the question to continue with the shrinkage system in a long stope. At this stage, the remaining 140 ft. of the stope could be emptied of broken ore, filled with waste to within 6 ft. of the back and turned into a regular square-set stope on top of the waste fill and carried up, from there on, as a regular square-set stope. This method would entail the maximum amount of timber expense, therefore a plan was considered to cut up this 140 ft. of stope into four short shrinkage stopes and work them rapidly one at a time. However, if the back of the stope should still, at any time, be dangerous enough to require timbering, the miners would have to work under dangerous ground to get the timbering started.

It was at this point that the plan for the present "slot" system was evolved. The drawing chutes in this stope were 20 ft. apart on the 200-motor level. The stope was emptied of broken ore and then, beginning at the southern end of the stope at No. 1 chute, square-set timbering was started on suitable stringers, so that the first set was directly over the chute. Sets were then added on the same horizontal floor elevation as the first set, until a single row of sets was completed from hanging wall to footwall and securely blocked. This represented the first floor of the No. 1 slot. The second, third and succeeding floors were added as rapidly as possible until the back of the stope was reached. The tops of the sets were blocked against the back for the time being, until the next slot timbering over chute 3 (40 ft. north) was completed. The inside faces of these two slots were then lagged off and the open stope between them was filled with waste to within 6 ft. of the back. The top of the waste was then floored over, and the south end of the stope, 40 ft. long, was ready to be mined according to the new plan. The two slots, over chutes 1 and 3, were then carried on up to the next level. Care was taken not to break the ground any wider than necessary to stand the square sets. Test holes were drilled into the footwall at 12-ft. intervals and occasionally into the well defined hanging wall, to make certain that

in each floor of the slot all the ore had been mined from hanging wall to footwall (Fig. 1).

To steady the square sets in these slots, they were occasionally tied with discarded 3/8-in. wire rope (Fig. 2). The ore in the pillar, or plug between the two slots was then mined by the shrinkage system to the next level.

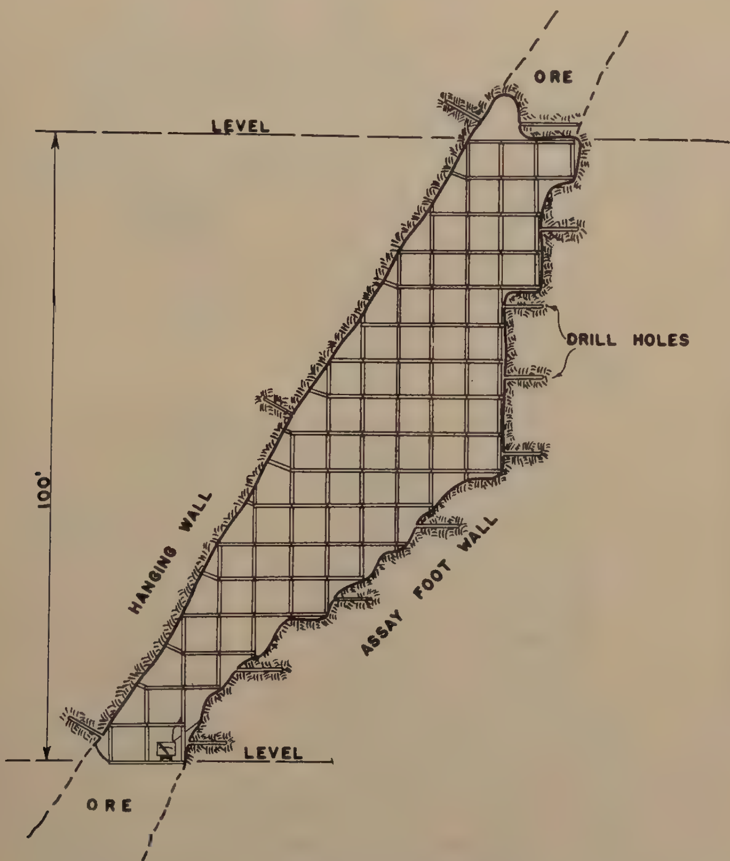

Fig. 1.—Method of prospecting footwall and hanging wall with drill holes as slot is carried up.

When all the broken ore was drawn from between the slots, waste fill was applied. The inside of each slot was lagged off, so that the slots would remain open after the 40-ft. stope was filled with waste. As the waste fill was put in, the slot timbers were occasionally cabled to "dead men" buried in the waste, to keep the waste fill from pushing or bulging out the slot timbers later on when the neighboring section was being mined (Fig. 2).

The chief advantage of the slot system of mining is in the saving of timber. Ground that might prove too dangerous to mine in a larger open-shrinkage stope can be cut up into shorter blocks by these timbered slots and the plug between the slots can be mined rapidly by the shrinkage system instead of all square sets. Or, if even the small section proves troublesome, it is comparatively easy to change over to a cut-and-fill method, after drawing out the already broken ore and filling with waste

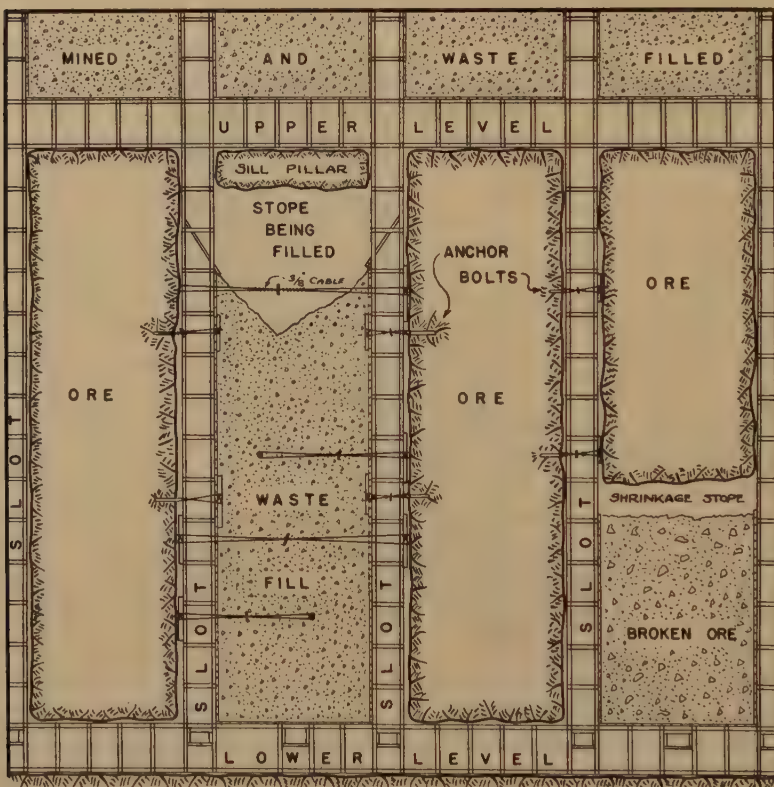

FIG. 2.—METHOD OF MINING PILLARS BETWEEN SLOTS AND TYING SLOT TIMBERS.

to within working distance of the back. In making this change in the operation, which was done on one or two occasions, the men have the safe, timbered slots to work in for their protection. With slusher machines and scraper equipment set up in the slot, the necessity for working under any bad ground is almost eliminated. If, after changing to the cut-and-fill method, the back or hanging wall persists in giving trouble, it is a simple matter to change to straight square-setting. Another advantage is in the ease with which waste fill can be put into the stope through the slots from the level above.

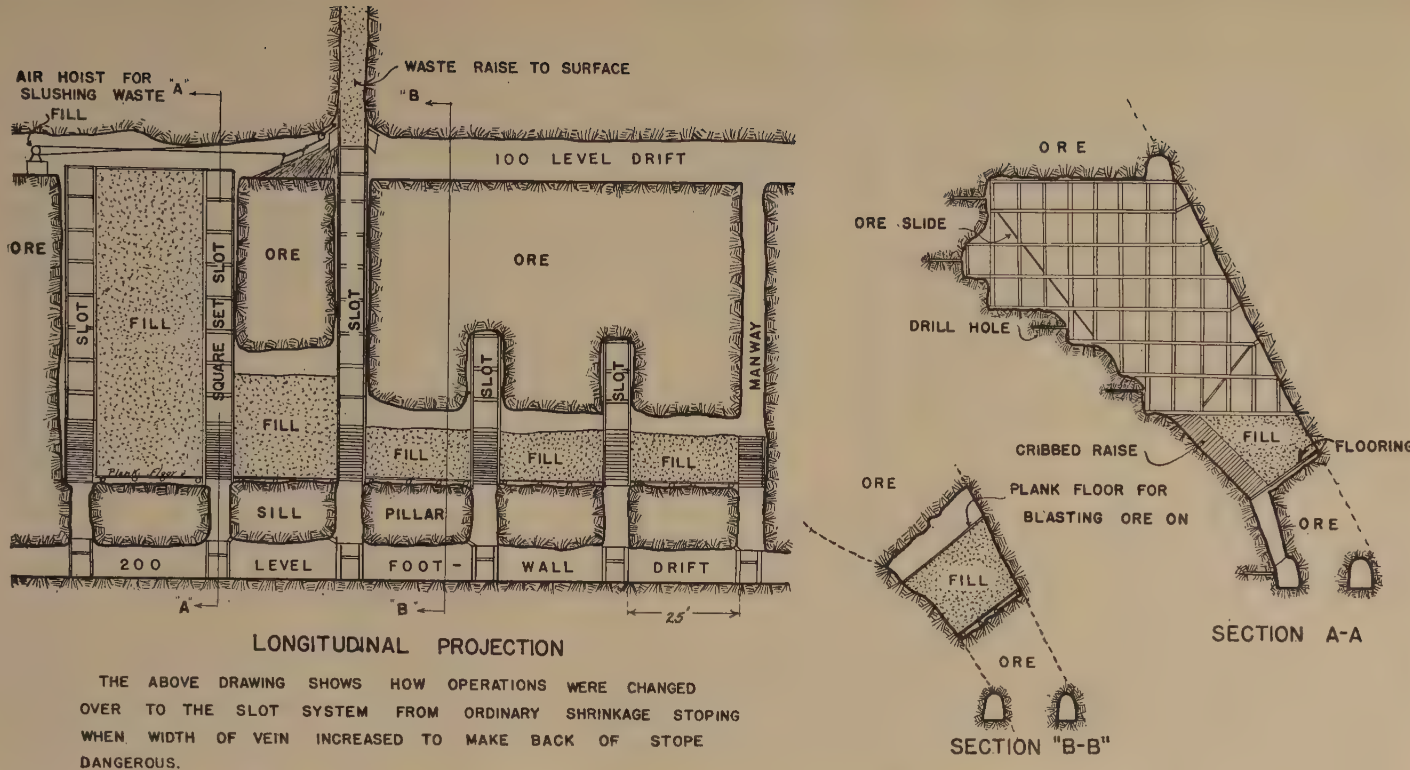

Fig. 3—Square-set slot and pillar method of mining as used at Silver Queen mine.

In carrying up an ordinary shrinkage stope, in a vein where the metal content fades out irregularly into the footwall, it is necessary to drill frequent test holes to make sure no ore is being left behind. This is troublesome in a contract stope, where the machine men are impatient to break ground as fast as they can and do not like to wait for assay returns from the footwall drill holes. Here again the slots prove convenient for this footwall drilling, and when completed the slots show a true section of the extent of the ore from hanging wall to footwall, all the way up to the next level; also the plug between the slots can then be mined as rapidly as possible without danger of leaving pay ore in the footwall of the stope.

The slots can be used for manway and ore chutes at the same time by lagging off one set next to the hanging wall as the slot is being carried up. The distance between slots is a matter for decision by those in charge of actual daily stoping operations. From 25 to 35 ft. between slots has proved about the right distance at the Golden Queen mine, for ore varying from 15 to 30 ft. in width.

The flexibility of this system is its strongest point, and from a safety point of view the results have been excellent, in spite of the fact that the system has been applied only to portions of the vein where dangerous working conditions were anticipated. The slots afford easy access for air and water lines, as well as convenient places for keeping machines, hoses, drill steel, etc. Good ventilation is also easily maintained. Only two miners per shift are necessary to carry on the entire work in a section, except for the tramming on the main level, and the necessary installation and upkeep of air and water lines.

DISCUSSION

(*E. D. Gardner presiding*)

C. F. Jackson,* Washington, D. C.—This illustrates the danger of jumping at conclusions as to what ground will do when in stopes and more ground is opened up on the basis of the way it stands in development openings. It seems to me that when the stopes were opened up and a considerable area of hanging wall was exposed, difficulties arose and it was necessary to adopt this system.

E. D. Gardner,† Tucson, Ariz.—I think that the principal advantage of this method is its flexibility. If the ground gets worse straight square-setting can be used, or if the walls improve the ore can be mined by shrinkage. I think it is well adapted for varying conditions which might be encountered in any part of the vein.

The old Silver Queen vein, as they called it in the beginning, had a tendency to roll out into the footwall. That condition developed in the first shrinkage stope that was mined. A roll was followed out and too much ground was opened up, permitting the ground to cave.

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* Chief Engineer, Mining Division, U.S. Bureau of Mines.

† Supervising Engineer, Southwest Experiment Station, U. S. Bureau of Mines.

W. R. Chedsey,* State College, Pa.—Fig. 3 refers to this same mine, even though the title says Silver Queen instead of Golden Queen.

E. D. Gardner.—I think that the people in the district call it the Silver Queen mine, but the Golden Queen Mining Co.

J. B. Canada,† New York, N. Y.—It seems to me there ought to be some figures of conclusive cost, because the slot system is based on a square-set cost basis and a mine that is opened up primarily for shrinkage stope and then has to go into a square-set system will naturally be more expensive. It might be found necessary to use a square-set system altogether.

There ought to be some discussion, perhaps in the future, on this paper as to the cost of the system that has been mapped out, and how far apart economically the slots should be spaced.

E. D. Gardner.—I think that is determined by the operating force when they start on any particular block of ore.

R. H. Dickson,‡ New York, N. Y.—There is a paragraph on this particular subject in Bureau of Mines *Information Circular* 6931, stating that the actual mining costs are less than a dollar a ton.

E. D. Gardner.—The ore runs around $30 per ton at present prices of metals, so the importance of not losing ore overshadows any extra cost that might be attributed to the timber.

G. Sherman, New York, N. Y.—It is quite often difficult to get costs from companies for one reason or another. In comparing costs between operations, it must be recognized that labor conditions, wages, etc., vary widely in different districts and, from year to year in the same district.

I think a paper presented on a mining method should make it possible to draw some approximate cost comparisons with former methods or other mines. It could be best expressed as tons per man per hour, or per day, a unit of performance recognized everywhere. If the number of pounds of explosives used, the number of board feet of timber per ton, and the number of tons per man in stoping is given, the situation is practically covered.

C. A. Kumke.—Experience is showing that the double slot is more satisfactory than the single slot. Ton for ton the double slot goes up faster, needs less cabling and tying and is steadier when against fresh waste fill.

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* Professor of Mining, Pennsylvania State College.

† Mine Superintendent, Dewey Gold Mines Trust.

‡ Mining Engineer, General Chemical Co.

Mining Methods and Ore Estimations at the Hog Mountain Mine

By N. O. Johnson,* Junior Member A.I.M.E.

(New York Meeting, February, 1937)

The Hog Mountain mine is a pyritic-gold property in the north central part of Tallapoosa County, Alabama, at an elevation of 800 ft. in the southern Appalachian region. It is 13 miles by a good secondary road from Alexander City, a station on the Central of Georgia Railroad.

Since its discovery in 1839, the mine has been worked intermittently. The records of past operations are few and are scattered over a long period of time; however, available information indicates a production of approximately 12,000 oz. of gold prior to the present company's activities, which started in 1933. Additional 7844 oz. has been produced under the direction of the present management. The ore from stopes has averaged 0.200 oz. gold per ton; the ore from development, 0.136 oz. gold per ton. Silver, copper, iron and sulphur are also present but to date are of minor importance. The grade of ore in comparatively narrow veins makes necessary extreme vigilance in the methods of mining.

Geology

Wall Rock.—Hog Mountain consists of a quartz-diorite intrusion approximately 5000 ft. long by 1200 ft. wide. Its resistance to weathering, as compared to the surrounding schists, has led to the formation of a conspicuous ridge rising 400 ft. above the adjacent country. All the mining operations are confined within the limits of this quartz-diorite formation. Except where there is excessive shearing, either parallel or at flat angles to the quartz veins, all the walls of the stopes within the limits of the veins are strong enough to make timbering unnecessary. Where shearing exists, there may be a sloughing off of the walls, especially in the process of drawing the ore. When this occurs, stulls are placed in the stope, either while the stope is advancing, or while ore is being drawn from a completed stope. In general, very little timber is required.

Veins.—The quartz veins have been deposited along fracture planes in the quartz diorite. There are 20 principal veins, all of which are roughly parallel and strike at right angles to the long axis of the formation.

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* Field Engineer, The Dorr Company; formerly Chief Engineer, Hog Mountain Gold Mining and Milling Co., Alexander City, Ala.

They all dip steeply to the north. The six veins in the northernmost part of the main plug have a limited amount of production because of the tonguelike formation in which they exist; however, indications show that the veins in the main body will extend to considerable depth. Present workings prove that the veins do not extend beyond the limits of the quartz-diorite formation. Past operations have been confined wholly within the tonguelike structure at the northern end of the formation but the main body of the plug is now being explored.

Ore Deposits.—In developing the tonnage shown below, 4761 ft. of drifting along the veins was required. Adjusting this total mine tonnage

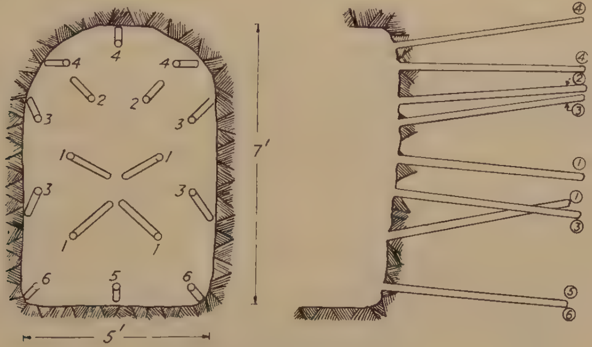

FIG. 1.—DRIFT HEADING.

to the corrected assays shows that the veins are 45 per cent productive of the past grade of ore sent to mill. Better grades of ore will reduce the productiveness of the veins, and vice versa.

	Tons		Tons
Mined from stopes	51,380	Unbroken ore (estimated)	25,000
Mined from development	10,785	Old workings (estimated)	5,000
Broken ore	3,442		
		Developed underground	95,607

This table shows that 40 per cent of the vein area developed may be of sufficient grade and width to warrant economic stoping. The individual ore bodies within the veins, although irregularly lenticular, usually have a rake to the east in accordance with the main plug. The lenses range from 100 to 250 ft. in length and extend vertically to the surface, which at present is 200 ft. Usually the veins are continuous between deposits, but the vein material is too narrow or too low in grade to mine. Parallel veins and stringers are numerous throughout the main body. The width of the deposits rarely reaches 20 ft. and generally in such widths the vein consists of multiple stringers separated by waste.

Ore.—The ore consists of stringers and massive intergrowths of pyrrhotite, pyrite and chalcopyrite in quartz fissures. The gangue mate-

rial varies from a typical granite to a highly silicified quartz diorite. The gold lies mainly in the pyrrhotite, though it is believed the other sulphides also contain small quantities of gold.

The average grade of the selected sulphides (mainly pyrrhotite) is 3 oz. gold per ton; that of the ore delivered to the mill (over-all average of 54,856 tons of development and stoped ore), 0.185 oz. gold per ton. The dilution is shown to be approximately 25 per cent, of which 11 per cent is removed by surface sorting operations. The following example illustrates more clearly the dilution problem:

Material	Tons	Assay
Ore	100	0.203
Waste	25	0.025
Ore mined and sent to mill	125	0.169
Waste sorted out (11 per cent)	13.9	0.025
Ore milled	111.1	0.185
Dilution, 11.1 per cent		

This dilution represents, approximately, the total amount of waste by-passed through a grizzly.

The development ore can be treated at a profit, disregarding the breaking and overhead charges that exist regardless of the working face. In stoping the ore, permissible dilution is a function of the final profits.

Development

The two veins on the extreme northern end of the quartz-diorite plug have been developed through an adit. All the major veins south of these two veins, worked from the adit level, are developed by a 200-ft. vertical shaft.

Levels are driven at 100-ft. intervals. Raises are driven in questionable areas to discover the possibilities of ore in the area. The completed raises are also used for ventilation and ore transfer. Two parallel crosscuts 300 ft. apart, one on the 100-ft. level and the other on the 200-ft. level, are driven to the south of the older working in the shaft. Occasionally short crosscut tunnels are driven off the main vein in an effort to locate parallel ore shoots.

Drifts 5 by 7 ft. are commonly used. The standard drilled round is illustrated in Fig. 1. Raises are driven 5 by 10 ft. for the first 15 ft. advanced, to allow for standard chute and manway. From that point upward, a 5 by 5-ft. raise is carried directly above the chute. To minimize damage to the ladders and the pipe lines, the ladders are made of poles, 3 in. in diameter at the large end. The poles are spaced 10 in. apart with 1-in. holes drilled every 18 in., in which are inserted discarded

pieces of drill steel to serve as rungs. Both the ladders and the pipe lines are held in place by wooden plugs driven in specially drilled holes. (Figs. 2, 3 and 4.)

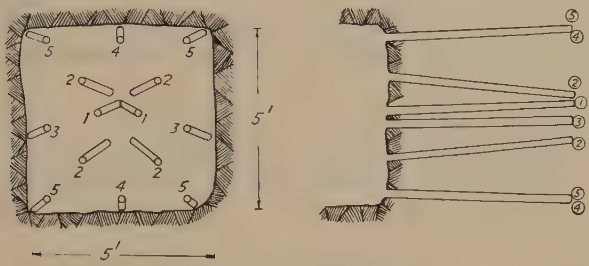

FIG. 2.—RAISE HEADING.

Practically all development work is done on company time. Contract work has been found to be less per foot in long raise headings, and for that reason raises between levels and to the surface are done under contract.

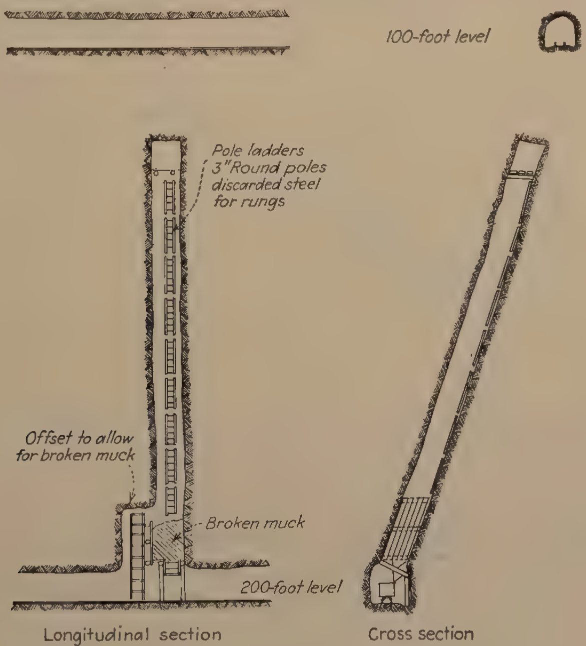

FIG. 3.—DEVELOPMENT RAISE.

The prices range from \$4.50 to \$5.00 per foot, depending upon the distance above the track level. Under this agreement the contractor pays for all the explosives, in addition to all the labor directly connected with raise.

Timbering because of bad ground has not been necessary in any of the headings. Contractors are guaranteed a minimum wage equal to the

regular company wage scale, and any additional amount earned on the contract is paid to them in the form of a bonus. In general, contract men average $1 per day more than company men, and for that reason contracts are given only to the best company men as an incentive for the new men starting to work in the mine. The cost per unit (footage or tonnage) is the same to the company, because of the increased production from the contract work.

Diamond-drilling has been experimented with, but because of the short time it has been used, no practical data have been gathered. The present development program calls for extensive drilling from both underground and surface setups at an over-all cost of $2.75 per foot.

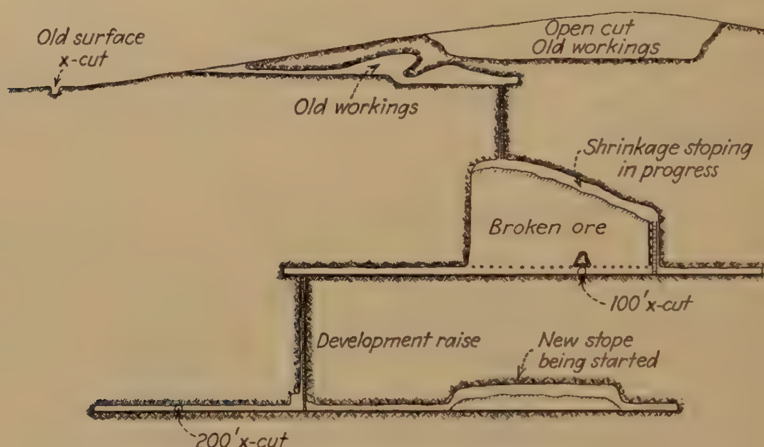

FIG. 4.—LONGITUDINAL SECTION OF TYPICAL HOG MOUNTAIN VEIN.

STOPING

Shrinkage stoping is used because of the firm walls and the moderate width of the veins. Chutes are placed at intervals from 12 to 20 ft., and the bottom of the stope is securely timbered and lagged; or if the bottom of the block is poor, pillars are left between the chutes, up to the limits of the ore. Usually, before stoping of a given block begins, a raise is driven through to the level above. This raise acts as a cut to break to in addition to the main purpose of exploring the area. If the raise is carried to the workings above or to the surface, it will aid materially in ventilating the stope.

Stoping machines are employed for drilling out the stopes, and drifting machines are used to advantage on horizontal holes. Mounted drills in the stopes are useful in the robbing of pillars in completed stopes. In using these machines, the column bars are braced horizontally between the walls of the veins, and from this setup long drill holes can be placed in any direction. About 40 per cent of the broken ore is withdrawn to

maintain the surface of the fill at a convenient level for drilling. Every effort is made to break all boulders and to support all slabs that might fall while the ore is being drawn, but in spite of every precaution, the ore arches over the chutes, requiring dynamite to dislodge it. Reducing the interval between chutes has greatly reduced the trouble of chutes hanging up; however, water from seepage and from the drill machines has caused the fine material in the stopes to cement together, and also has caused trouble while drawing. Mucking in the stopes is necessary occasionally to clean off the loose material hanging up on the footwall of the vein.

Records for August 1936 illustrate comparative figures in three new representative stopes (Table 1).

TABLE 1.—*Records at Hog Mountain Mine for August* 1936

	No. 11 Stope	No. 15 Stope	No. 16 Stope	Total and Averages
Broken ore, 8/1/36	262	None	None	262
Broken ore, 9/1/36	1,441	643	95	2,179
Tons broken during August	1,457	1,072	271	2,800
Tons trammed	1,119	354	176	1,649
Drill shifts (18 man-hr. per shift)	104.5	67.6	21.9	194
Timber shifts (9 man-hr. per shift)	37.9	72.8	48.0	158.7
Mucking shifts (9 man-hr. per shift)	112.5	40.2	20.4	173.1
Holes blasted	1,055	707	228	1,990
Sticks dynamite used	8,610	5,580	1,420	15,610
Labor cost	$778.72	$554.74	$247.84	$1,581.30
Supply cost	$437.26	$301.71	$85.80	$824.77
Tons trammed per mucking shift	10.0	8.8	8.6	9.5
Tons broken per drill shift	13.9	15.9	12.4	14.4
Sticks dynamite per ton broken	5.92	5.20	5.24	5.57
Holes per drill shift	10.1	10.4	10.4	10.2
Tons broken per hole	1.38	1.52	1.19	1.41

Drilling

Drilling in the stopes is mostly done on company time. Contract drilling has been tried in the stopes, with the drillers receiving approximately 5¢ per foot of hole drilled, depending upon the hardness of the rock and the location and condition of the stope. The average cost per foot of hole for labor (drillman and helper) is 10¢. This method for increasing the drilling footage has been abandoned because of the close supervision required in placing the holes. To date the most successful method for increased output in the stopes is the tonnage system, in which the stope, after being prepared by the company, is turned over to a contractor who is paid according to the tonnage broken. Every 10 ft. of vertical advance made in the stope is systematically sampled, and the width of the prevailing vein in the stope measured. With the samples

taken at 10-ft. intervals along the horizontal and every 10 ft. along the vertical, a perfect cross section of the stope is obtained. The contractor is paid only for the width of the vein, thus preventing dilution of the ore and overbreaking, which the contractor might attempt to do for increased tonnage. The contractor is responsible for all labor, including the mucking of the shrinkage, and furnishes his own explosives. He does not have to keep up the necessary timbering or to remove the broken ore after the stope is completed.

Two stopers are used continuously in each stope, two are in constant use in development raise headings, and one is always in the shop as a spare.

TABLE 2.—*Stoping Data*[a]

Rock	Number of Bits Used	Number of Holes	Footage	Inches per Bit
Granite	38	11	55	17.5
Granite	42	17	91	26
Granite	15	6	30	24
Quartz	42	8	50	14.3
Quartz	52	13	65	15
Quartz	9	3	15	20
Quartz	57	17	85	18
Granite	42	17	85	24.3
Granite	34	15	83	29.3
Quartz	56	14	56	12
Granite	30	15	83	33.2
Quartz	35	7	42	14.4
Quartz	51	9	62	14.4
Quartz	48	8	57	14.3
Quartz	60	10	71	14.10
Quartz	41	7	48	14
	652	177	978	
Average for bit in granite, in				25.5
Average for bit in quartz, in				14.7

[a] On last five days, used same 7 pieces of steel, 1-ft. changes; therefore, seven different sizes of bits. The 652 bits used were both new and regrinds. Stopers could probably be used with steel in 18-in. changes. Drifters could probably be used with steel in 24 to 30-in. changes.

Compressed air is supplied under 100-lb. pressure by one angle compound compressor and one tandem compound compressor. The capacity of the two combined is 1500 cu. ft. of free air per minute. The air is piped to the mine through a 4-in. main; 3-in. laterals are used as feeders on the two levels. There are two air receivers on the surface and one on the 200-ft. level, having a combined capacity of 576 cu. ft. Water under 140 lb. pressure is piped to all the drills.

The machine drills are sharpened in one oil-fired furnace. All work is done on two shifts, with the combined daily average of drills sharpened being approximately 1000. The cost of sharpening averages 2.7¢ per drill.

Detachable bits have been experimented with, and show a substantial saving, particularly in the stopes and the raises, where the time lost in transferring the steel is considerable. The total comparative costs arrived at by two weeks of intensive study is 11.47¢ per foot of hole drilled with detachables against 12.70¢ per foot for the same items with shop bits. Table 2 illustrates some of the compiled data.

Timbering

Native timber is used exclusively. Delivered to the mine, pine and oak cut at the near-by sawmills are $12 and $14 per 1000, respectively. Round oak poles are used for stulls; posts and caps averaging 8-in. dia. are cut and delivered for approximately 2¢ per linear foot.

All the round timber used for supporting the broken ore in the stopes is framed by a company timber crew. About 655 man-hours are required to completely timber a stope 100 ft. long, before the actual stoping operations can begin.

Tramming

All the tramming is done by hand in 16-cu. ft., rotary, end-dump cars. The average tramming distance from all the working faces and the stopes to the shaft station is 800 ft. The number of cars is counted by the muckers and checked again by the hoistman on the surface.

As the haulage distance increases, mechanical trammers will be installed on the main crosscut tunnel on the 200-ft. level. The ore from the 100-ft. level workings will be transferred to the 200-ft. level through raises. Waste from the 100-ft. level is dumped into completed stopes. The completed stopes are filled whenever convenient, as there is no danger of the unsupported walls caving due to bad ground.

Hoisting

All of the present production is hoisted through a 200-ft. shaft located between the Barren and Blue veins at the northern end of the main plug formation. This shaft is capable of handling the production for the immediate future, but is likely to be abandoned as the main part of the plug, which is to the south, is developed.

The shaft is timbered with standard shaft sets, spaced 5-ft. centers, and divided into two 4½ by 6-ft. compartments. At present only one compartment is used for hoisting, the other being a ladderway and entrance for all pipe and electric lines. The cage is of local design and runs between 4 by 6-in. pine guides.

Switching facilities on the 100-ft. and the 200-ft. levels, as well as a double track on the surface leading to the ore bin, minimize the lost time underground waiting for the cage. An ore pocket between the 100-ft. and the 200-ft. levels, 50 ft. from the shaft station, is drawn for 2 hr. between shifts by two men, one drawing the chute while the other cages the cars, during which time a maximum of 50 cars can be handled.

The cable, ⅞ in. in diameter, is wound on a 36-in. dia. drum. This single drum hoist is geared to a 50-hp., 220-volt induction motor. The hoist is operated for two 9-hr. shifts, and for 2 hr. between shifts per day. During this total time it handles on the average of 200 cars, in addition to the men and supplies. Hoisting and surface dumping represents 6.7 per cent of the total mining cost.

Sampling and Estimating

In sampling development work, dilution is accepted as an unavoidable evil. Samples are cut over the full width of every quartz vein in the working face. Fifteen per cent wall dilution is added to all narrow veins. This accounts for the efficiency of the pickers on the sorting belt and for the amount of fines that are by-passed through the grizzly. If the quartz obtained from the face sample has sufficient metal content to maintain 15 per cent of the total wall rock present at a development ore grade, the muck broken from that heading is sent to the bin; if not, it is considered waste. Widths and assays are recorded in the engineering office on a daily sampler's report.

All drifts and raises are sampled after each round, generally at 5-ft. intervals. The samples are taken at the beginning of every morning shift before the drillmen have started their actual drilling, or while the muckers are cleaning out the heading blasted by the night shift. In double-shifting one heading, samples are taken once every two rounds. The backs of the stopes are sampled each 10 ft. advanced and at 10-ft. intervals along the fill in the stope. This method gives as near a perfect cross section of the stope as possible, both for widths and for assays. It is customary to take grab samples in the stope because of the time involved in getting face samples. These grab samples are characteristic of the vein in the back, for the stopes are carried only within the walls of the vein.

The width of the vein and the distance from the end of the stope is recorded for each sample. The stope maps, together with the assays and the widths of these grab samples, are used in calculating the reserves of broken ore. Grab samples, consisting of two double handfuls, are taken from each car of ore and waste as it is dumped. The composites of the hand samples from each stope and development headings are assayed daily.

Assay Map.—Separate assay maps are used for the plan and the section. The different levels are shown in the plan with the widths and assays of the face samples on one side of the drift, and the corresponding

muck samples on the other side. Longitudinal sections are made of every vein, and the various stopes are inserted in their proper location. The assays and the widths of every grab sample are then plotted in their respective places. Vein splits or other irregularities are noted on both the plan and the longitudinal maps.

Tonnage and Assay Report.—The assay of the grab samples and the record of the cars trammed furnish the basis for the monthly tonnage and assay report. Cars and assays from every stope and development heading are recorded daily, and at the end of the month the sum and average are reported for individual workings and for the whole mine. The actual tonnage milled plus the tonnage sorted out as waste furnished the total tonnage delivered to the jaw crusher, which, when divided by the number of cars dumped in the jaw-crusher bin, furnished a factor from which to calculate the actual tonnage from each stope and heading. With the

TABLE 3.—*Vein Productiveness and Production*

Developed ore, tons	95,607	Ore milled, tons	54,856
Lateral work along veins, ft.	4,761	Average mill heads, oz. per ton	0.185
Productiveness, tons per ft.	20.8		
Total lateral work, ft.	7,225	Average daily production, tons per mill-operating day	111.5
Productiveness, tons per ft.	13.2		
Period covered, months	31	Operating time, per cent.	53.3

corrected tonnage plus the average assay value, the gross value of the ore from each stope can be tabulated from month to month until the stope is finally exhausted, thus a fairly accurate record of its production is available.

The factors used for correcting the tonnage and metal content of the ore furnish interesting figures on the accuracy of counting cars and of grab sampling. The tonnage factor seldom varies more than a few per cent on either side of 0.75 ton per car. Since the cars accurately hold ¾ ton, the tonnage record can be regarded as very accurate. The gold content is usually a little low. For the past two years figures indicate that the mine assays are roughly 11 per cent high, which is not considered unsatisfactory for grab samples. The discrepancy is probably due to the tendency of the mineral to break fine, and the natural inclination of the trammers to take the fine ore rather than the coarse ore in their grab samples. Perhaps the subconscious desire to secure a good sample may contribute to the discrepancy.

Calculation of Ore Reserves.—Ore reserves are estimated on the first and fifteenth of every month. The backs of all the stopes are surveyed on those dates and plotted on the assay maps. The broken ore is measured to obtain the cubic content, then divided by 20 (cubic foot per ton factor), giving the broken tonnage in each stope. The length and the height of the stope can be readily obtained by measurements, whereas the width is obtained from the average width of all the grab samples recorded

TABLE 4.—*Distribution of Mining Costs*
(DEVELOPMENT EXCLUDED)

	PER CENT	PER CENT
Breaking ore (shrinkage stoping):		
Labor	22.1	
Explosives	14.7	
Drills	24.9	61.7
Tramming and mucking		13.0
Timbering		7.3
Hoisting and dumping		6.7
Assaying		2.3
Other expenses		9.0
Total		100.0

TABLE 5.—*Underground Efficiency*

	MAN-HR. PER TON
Mining (shrinkage stoping):	
Breaking ore	1.85
Tramming and mucking	1.00
Timbering	0.18
Hoisting and dumping	0.22
Assaying	0.06
Other expenses	0.31
Total stoping labor	3.35
Explosives, lb. per ton	1.5
Timber (excluding native poles), bd. ft. per ton	1.2
Power, kw. per ton	34.0
Drifting:	
Man-hours per foot advance	9.7
Explosives, lb. per ft	12.2
Advancement per shift, ft	3.9
Raising:	
Man-hours per foot advance	10.3
Explosives, lb. per ft	9.5
Advancement per shift, ft	3.0
Over-all labor (surface and underground):	
Man-hours per ton material handled	6.33

in that stope to date. An adjustment is made for pillars and other voids known to exist at the time of the measuring.

If the block of ore is developed on three sides, it is considered suitable as a stoping area. The exact limits of the stopes cannot be determined until the stoping operations are completed; however, for calculating the unbroken ore available, it is customary to consider the ore extending 30 ft. in the drifts and crosscuts.

The full assay of all samples is employed. The modification of high assays is warranted where the determination of the reserve is based on only a few samples, but taking several thousand samples, abnormally

high assays will be offset by those abnormally low, and the average will more likely be correct if the high ones are not modified.

Cost and Stope Statements

The tonnage and gross value of the ore from the various stopes are taken from the daily operating report. The number of shifts worked in each stope by machine drillers, timbermen and muckers is taken from the daily time cards; the amount of powder, caps, fuse and timber is taken from shift-boss reports, and all distributed to the various stopes. Development costs have been distributed on the same basis, but as there is little or no variation from day to day, this has been discontinued. Using this method, each stope has an individual record at the end of every month. It is believed that this method shows, much better than mere assays, which stope requires special attention and when it should be stopped.

No pretense of extreme accuracy is made for this report. It is based on many approximations. It must be drafted by an engineer thoroughly familiar with the operations of the mine, and he must exercise his judgment continually in distributing the various charges. This close touch with changing conditions underground renders more valuable information to the operating personnel than waiting for the regular monthly financial report in which all stoping operations are lumped together.

This method of computing the costs of each stope furnishes accurate records of the labor and supplies for each stope, and from experience these seem to be approximately 60 per cent of the total cost of the ore.

The so-called indirect charges, representing the balance of approximately 40 per cent, cover: drill steel and sharpening, drill maintenance, assaying, sampling and surveying, supervision, ventilation and pipe lines, compressed air, hoisting, pumping (camp and mine water), supplies (carbide, oils, shovels, picks, etc.), track and miscellaneous repairs.

While the direct charges are almost directly proportional to the tonnage mined, it can be seen that the indirect charges would not increase in proportion to the tonnage, and for an increased output these should be less than 40 per cent.

Data on production, distribution of mining costs, and underground efficiency are given in Tables 3, 4 and 5.

Mining Practice at the Bell Limestone Mine

By Samuel M. Shallcross,* Member A.I.M.E.

(Pennsylvania State College Meeting, September, 1936)

The principal function of the Bell limestone mine, of the American Lime & Stone Co., at Bellefonte, Pa., is to supply raw material to the company's modern rotary kiln plant at Bellefonte. Because mining costs are naturally higher than many open quarries with more favorably lying strata, it cannot compete, except to a limited extent, in the markets for road material and fluxing stone.

In the broad sense, mining does have an advantage over the average open quarry when the end products are designed for and controlled by process and final inspection for the exacting demands of the industrial and chemical markets. Approximately 80 per cent of the output of the Bell mine is sold in the industrial and chemical markets. And as higher standards are set each year for construction and agricultural work, the remainder of the mine output goes to these two classes. Mining produces more fines than open quarries, and these fines are difficult to market; on the other hand, owing to absence of contaminating clay and impure rock, a greater percentage of the output of a mine is marketable. Because of the great production of fines, rotary kilns for lime burning are necessary when the stone must be mined.

In 1921 the American Lime & Stone Co. operated 19 quarries and 72 lime shaft kilns spread over an area of 40 miles between Bellefonte and Altoona, Pa. At that time the lime-production capacity of the 72 kilns amounted to approximately 700 tons per day. After 15 years the company operates three rotary kilns and six shaft kilns, with a combined output of approximately 500 tons of lime per day, concentrated in its single Bellefonte plant (Fig. 1). Its only stone-producing plants are now the limestone mine at Bellefonte and a construction-stone open quarry at Union Furnace, near Tyrone, Pennsylvania.

Since the middle of 1922, the management of the American Lime & Stone Co. has been under the direction and ownership of the Warner Company, of Philadelphia. To the Warner Company's experience, gained through successful operation of lime plants in eastern Pennsylvania for many years, and the successful solution of material-handling problems

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at its Delaware River sand and gravel plants, should be given the main credit for the successful consolidation of the American Lime & Stone Company's many plants and the development of the Bell mine.

Geology

Mining operations are confined to the high-grade limestone beds locally called the "Bellefonte ledge," but properly known as the Lowville

Fig. 1.—Plant of American Lime and Stone Company at Bellefonte, Pa.

limestone of the Black River group, Ordovician Period. The "Bellefonte ledge" outcrops on the southeast side of Bald Eagle Mountain, in Centre County, Pennsylvania. It can be traced from Jacksonville, Pa., 12 miles northeast of Bellefonte, to Tyrone Forge, Pa., 30 miles southwest of Bellefonte. Naturally, the "Bellefonte ledge" varies in thickness, dip and even impurities, but in the vicinity of Bellefonte it is remarkable for its uniformity of structure and chemical composition. Its dip varies from 45° to vertical. It is approximately 70 ft. thick in the Bellefonte area. Such extreme dips compel mining methods, because of the limited amount of high-grade stone that can be obtained by open quarrying.

Mining

The sinking of the inclined shaft was started in June, 1921, 80 ft. south of the footwall of the "Bellefonte ledge." Crosscuts were driven into the good stone. The shaft is 8 by 24 ft., thus providing three compartments. Two compartments are used for the balanced stone skips and the third for the man cage, ladder, water, air and power lines.

The dip of the ledges amounts to 52° at the surface but gradually increases to 56° at the 400-ft. level. Below the 400-ft. level the angle increases sharply until it reaches a maximum of 80° at the bottom of the shaft. (See Table 1.) At this point, however, the curve apparently turns back and the dip gradually flattens out. The shaft follows the dip of the strata.

Because of the thick and solid ledge above the shaft, timbering is not necessary except at one point between the 400 and 600-ft. levels.

TABLE 1.—*Inclination of Dip at Various Elevations*

	Sea Level Elevation, Ft.	Slope, Deg.
Surface	781	52
1st level	712	53
2nd level	649	55
3rd level	588	55½
4th level	525	57½
Old pocket	439	60½
600-ft. level	253	68½
Shaft bottom	92	80

At first the method of mining was that of overhand-underhand stoping. In 1923, however, the shrinkage-stope method was started and the overhand-underhand stoping gradually discontinued. Of the original four levels, the first and third levels were abandoned and the shrinkage stopes were carried from the second and fourth levels up through the first and third, respectively (Fig. 2). Shrinkage stoping, in spite of its heavy inventory of broken rock in the stopes, has proved to be more economical than the overhand-underhand method, and, what is more important, is capable of far greater tonnages. Even if power shovels had been purchased originally, as was contemplated, the eight-room limitation on four levels would have made very difficult the present production of 1200 to 1500 tons per day.

Shrinkage Stoping

As practiced on the second and fourth levels, the stopes are 300 ft. long on the strike of the vein. With 40-ft. vertical pillars, the net length of the stopes is 260 ft. The width is approximately 50 ft. at right angles to the dip, thus leaving a 20-ft. pillar on the hanging wall, to hold it up and prevent the shaly stone from contaminating the good stone.

Horizontal pillars 40 ft. wide protect the working levels. When production on the second and fourth levels was discontinued, the floor pillar on the second level was robbed. The fourth-level pillar will always be retained, as that level is used for drainage and pumping.

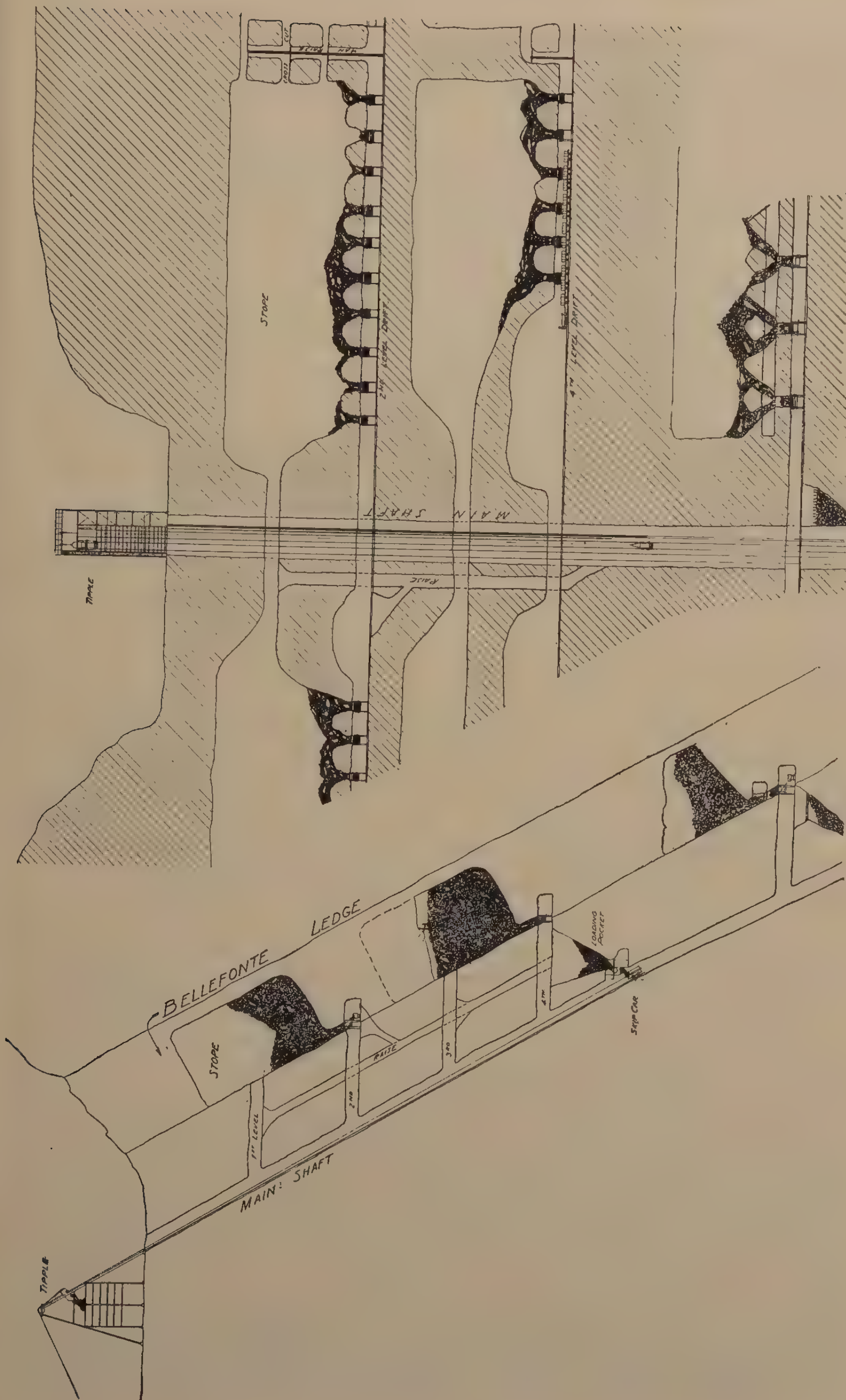

FIG. 2.—SHAFT AND LEVELS AT BELL MINE.

The sequence of operations as practiced on the second and fourth levels was as follows:

Drift.—Main haulage drifts, 9 by 11 ft., were driven east and west from the shaft cross entry. The lower southern corners of the drifts cut slightly into the footwall. Drifts are driven along the footwall so that the loose stone in the stopes above will draw most advantageously. Double drifts are driven every two or three stopes for track run-arounds.

Raises.—The stone or chute raises beneath each stope are on 25-ft. centers, on the footwall. Man raises are located in every other vertical pillar, and crosscuts are made into the stopes on both sides of each man raise. On the fourth level, every second or third man raise is carried up to the second level. On the second level, three man raises are carried to the surface for emergency exits and ventilation.

After all raises are completed, the chute timbers are set in place and the chute raises continued upward and longitudinally until they intersect, when stoping begins.

Mine Balance.—As is well known, loose stone weighs only approximately 60 per cent as much as solid stone, for a given volume. As the stone is shot down in the stopes, 40 per cent of the tonnage shot down must be removed through the chutes in the main drifts. This may be called the shrinkage stage. The trammers below must be careful not to draw those chutes directly beneath the point where the miners are working in the stopes above. When the stope has been carried to its maximum height, the stoping miners move forward into the next stope, which has just been completed for them by the development crews. If the development crews have just completed the development work of the next stope, then it may be said that they are balanced with the stoping crews, but in figuring the mine-tonnage balance, another test must be made.

Theoretically, the mine might be said to be in balance when the stoping miners have just completed a stope and are ready to move forward into a new stope that has just been completed by the development crews, who in turn start the development of an entirely new stope. Assuming that each miner in the stope shoots down 150 tons of stone per day, the mine tonnage will be made up of 40 per cent of the miner's production, which must be shrunk from his working stope. The balance of the mine's tonnage will be 60 per cent of the miner's production, and this 60 per cent will be obtained from the completed stope that the miner has just left. To withdraw more than 60 per cent of miner's production from completed stopes reduces the stone inventory, and a withdrawal of less than 60 per cent from completed stopes increases the stone inventory. According to this theoretical assumption, the minimum inventory of stone in stopes would be limited to the tonnage in one completed stope. On the second and fourth levels the average height of stopes was 100 ft. along the dip, therefore the solid block of stone mined from a stope was

approximately 100,000 tons. From a practical standpoint, however, with a demand of 1200 tons per working day, it was found that the necessary stope inventory required varied from 250,000 to 300,000 tons—equivalent to nearly a year's supply. This excess of inventory over theoretical can be explained as follows:

1. A minimum of eight miners in stopes is required for a production of 1200 tons per day. Since the width of the stopes does not permit more than three miners to work at a time, at least three stopes had to be working in the shrinkage stage. Mathematically, the minimum inventory in three stopes in the shrinkage stage is equivalent to the inventory of a completed stope.

2. Several stopes in the drawing stage had to be carried because the difficulty in drawing stone through the stone raises into the tram cars required a large number of working places.

3. Production factors of safety account for the balance of the inventory. It requires about nine months to increase the mine production rate and keep it in balance. Also, development was pushed ahead and heavy inventories maintained, to provide for emergencies such as the occurrence of water courses or caves.

Development Capital Costs

Shrinkage stoping naturally requires heavy capital outlays. Without counting the capital cost of headframe, hoist, compressors, and other surface necessities, and likewise eliminating the cost of tramming equipment underground, the shaft-development capital cost will require $150,000. This is heavy in first cost, but counting the amount of stone that can be recovered on each level is not a heavy charge against the production costs.

In addition to the shaft capital cost, there is likewise the stope development and inventory values, which add an additional $75,000, bringing the total development capital requirements to $225,000.

Tramming

Because of the short hauls and ideal tramming conditions, storage-battery locomotives are used. The rail used underground is 40 and 60-lb., and the grade is standardized at 0.5 per cent with the load. The 3-yd. Granby self-dumping mine cars with roller bearings (Fig. 3) are operated in trains of 10 to 12 cars. From the stope chutes (Fig. 4) the stone is brought to the dumping grizzlies near the shaft. "One-man stone" is required by the company's lime shaft kilns and, therefore, the grizzlies are spaced with 11-in. clear openings for this size. Beneath the grizzlies on the 600-ft. level are two large pockets holding 800 tons each. From

these accumulation pockets, the stone flows by gravity to the skip-loading machinery.

Fig. 3.—Cars dumping into loading pocket on 600-foot level.

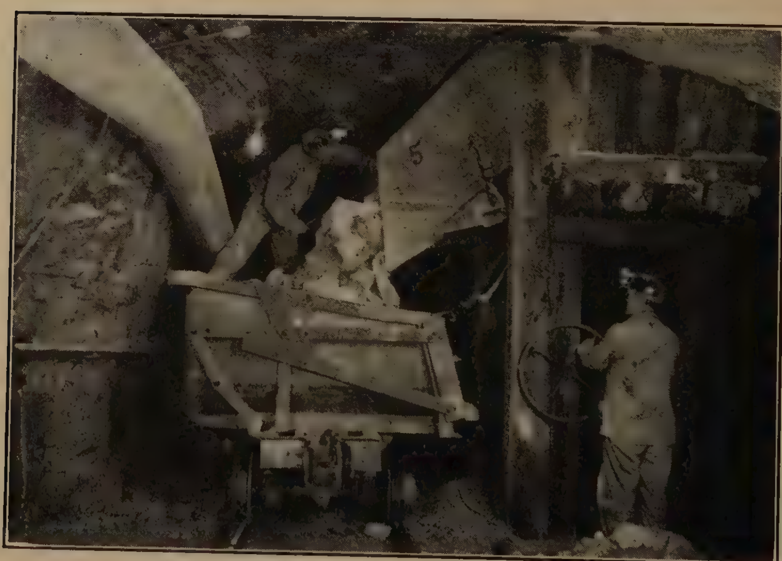

Fig. 4.—No. 5 chute east, 600-foot level.

Six-hundred-foot Level

The fundamental reasons for abandoning the second and fourth levels and developing the 600-ft. level were due to lack of foresight and

skimping on capital expenditure, both of which were only natural at the time the mine was first started. The principal differences between the 600-ft. level and the older levels are:

1. Stopes are carried up 200 ft. on the dip, thus halving the development costs (except for the cost of the bulldozing drift). (Fig. 5.)

2. A bulldozing drift is added above the main haulage drift in order to speed up the loading of cars at the chutes.

3. The large storage pockets above the 600-ft. loading pocket save in operating costs by preventing delays at one point from affecting the rest of the mine.

FIG. 5.—No. 3 STOPE WEST, 600-FOOT LEVEL.

It is practical to carry the stopes to a height of 200 ft. in the clear because of the steeper dip below the 400-ft. level. The "path of maximum descent" in broken solids in stopes, bins, chutes, etc., will always follow a vertical line down to the point of withdrawal. Therefore, if a line is drawn from the point of withdrawal vertically upwards until it meets the hanging wall, and then is continued upwards along the hanging wall to the top of the pile of broken stone in the stope, it will show in reverse the line path down which the stone will flow with maximum speed, and, therefore, quantity.

We can visualize the phenomenon further by imagining a cone inverted above the point of withdrawal; its vertex at the point of withdrawal; its vertical axis coinciding with the vertical above the point of withdrawal; its sides making an angle of 25° with this vertical axis; and its base at the top of the pile of broken material. Such a cone represents the material that will actually move during the shrinkage stage. Mate-

rial outside the cone will never move so long as the stope is kept filled. Material inside the cone will move with greater and greater speed as its position approaches the vertical axis of the inverted cone[1].

The practical miner, in speaking of this phenomenon, says: "The stone draws on the hanging wall and hangs up on the footwall." The shrinkage-stope method of mining is not so well adapted to dips less than about 65°. The fourth-level stopes drew better than the second-level stopes because the dip was slightly greater. To the west of the shaft, the surface terrain rises 100 ft. or more. But the second-level stopes could not be carried as high as desired, because the material on the footwall would not draw. These stopes, however, were carried up as far as practical by abandoning the stone along the footwall and continuing the stope upward on a narrower width. In some cases, as the stope was carried upward holes were drilled in this abandoned stone; then when the stope was completed and the stone drawn down, uncovering these drill holes, the stone was recovered. This was dangerous work because the roof above the men loading and firing these holes was too high for the men to trim.

Bulldozing Drift.—On the 600-ft. level the development of a bulldozing drift above the main drift requires only five stone raises per stope from the main drift up to the bulldozing grizzlies. Above the bulldozing drift 10 inclined raises are carried to the stope above. From these 10 raises the stone flows a small distance on to the grizzlies at the angle of rest, where it is easily controlled by men known as bulldozers. By block-holing and sledging, the large pieces are reduced to approximately "one-man size" and fall through the grizzlies into the main level chutes, where they are removed by the tramming crews. Bulldozers working under stopes in the shrinkage stage are instructed each day by the mine captain as to which chutes they are to draw from, so that the stope above will be properly drawn.

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[1] The author offers no apology for introducing in a descriptive article that which may seem to many a simple problem in elementary mechanics. The fundamental law or laws governing the flow of granular materials in bins is not now well known because no person or literature has been found that can or has treated the subject authoritatively. The author is now directing some simple experiments that should throw considerable light on the subject, but probably will have to leave to others the mathematical derivation of the fundamental law. The angle of 25° of the sides of the cone from the vertical axis and the later reference to 65° from the horizontal was made because these angles are functions of the angle of rest, and past experiments indicate that the angle of rest is a controlling factor. The importance of this law to engineers and industry should not be underestimated, because it is of vital importance to everyone that has anything to do with mining, milling, design of tanks or bins, closed-circuit grinding, segregation in bins, uniform withdrawal of material from bins, selective chutes, the grinding action of material moving in bins, stopes, and similar problems.

Drainage

Most of the water in the mine is surface water. A small mountain stream about ¼ mile east of the shaft was flumed over the outcrop a number of years ago, with very satisfactory results. At the present time plans are being drawn up to flume a larger stream that flows across the outcrop about ½ mile to the west. Tests have proved conclusively that this western stream is the principal offender in flood seasons. Fluming this stream will not only give a larger factor of safety to the pumping capacity in flood seasons, but also will save considerable pumping costs in dry seasons.

Water is not a problem at Bell mine. Normally the seepage amounts to 150 gal. per minute, whereas the pumping capacity is 3000 gal. per minute. On the fourth level, the main pumping level, the two pumps have a capacity of 1000 and 2000 gal. per minute, respectively. At the present time the 1000-gal. pump on the 600-ft. level pumps into the fourth-level sump. However, the seepage on the 600-ft. level is so small that its 100,000-gal. sump is pumped out only once a week. If water should become a problem, a high-lift pump would be installed on the 600-ft. level to pump directly to the surface, and the present pump would be moved from the 600-ft. to the fourth level, after that pump and the present fourth-level 1000-gal. pump had been converted to 2000-gal. pumps by installation of larger impellers and motors.

Ventilation

Until the 600-ft. level advances to a point where one of the man raises can be brought up to the fourth level, a small fan on the fourth level forces air through Ventube to the 600-ft. working places. With four openings to the surface and connecting raises between the second and fourth levels, natural ventilation provides large volumes of air, because of the chimney effect of the elevations of the surface openings. The westernmost air shaft has an elevation of approximately 140 ft. above the collar of the shaft. Because of relative temperatures and humidity of the outside air and the mine air, the direction of air flow in this western air shaft may be either up or down, but in either event provides fresh air far in excess of the mine requirements.

Lighting and Safety

All drifts and working places except the actual working faces are brilliantly lighted with high-power electric light bulbs. Carbide miners' lights are used at the working faces and are carried by miners working under the electric lights for use if the electric power fails.

Since the 600-ft. level has a life of 30 years at present outputs, air lines, water lines, power lines, battery-charging stations and stockrooms—in fact, all equipment and methods—have been designed for permanency.

The safety record of the Bell mine has been an outstanding one. Year after year it has stood high in the Bureau of Mines safety contests. Its safety record has been far superior to the company's record formerly with open quarries in the Bellefonte area.

Summary

1. Started in 1921, the Bell limestone mine of the American Lime & Stone Co. has completed 15 years of successful operation.

2. Located in the sharply dipping ledges of the Lowville limestone of the Black River group, the shrinkage-stope method of mining appealed to the management as the most economical, dependable and safe method of extraction.

3. The original overhand-underhand method of mining was soon abandoned for the more economical and safe shrinkage-stope method, which has been brought to a high stage of development on the new 600-ft. level.

4. The management pays particular attention to mine-production balance, and to the fundamental physical factors upon which the shrinkage-stope method is based.

5. Drainage and ventilation, although minor problems, have been provided for as though they were major problems.

6. Because of the long life of the working levels, permanency of equipment and mining methods, as well as safety factors and lighting, have en given major consideration.

DISCUSSION

(*Ross C. Purdy presiding*)

W. M. Weigel,* St. Louis, Mo.—How about the analysis of this rock, is it quite uniform?

S. M. Shallcross.—Yes.

A. W. Worthington,† Pittsburgh, Pa.—How do the fines compare in amount with those made in open-quarry operations?

S. M. Shallcross.—The underground operation results in a great deal more fines, probably double, which explains why we have had to use rotary kilns. The greater amount of fines may be due to grinding of the material on itself as it is drawn down through the stope as well as to heavier shooting.

D. W. Ross, Washington, Pa.—Can you estimate which of these factors is more important?

---

* Mineral Technologist, Missouri Pacific Railroad Co.

† Vice President and General Manager, Pittsburgh Limestone Corporation.

S. M. Shallcross.—No.

P. M. Tyler,* Washington, D. C.—Were there any records as to relative amount of explosives used? I realize that this is a rather small factor in cost production but if comparative data were available, they might afford some indication as to the relative increase in production of fines due to blasting.

S. M. Shallcross.—The open-cut and underground operations differ so greatly that comparative data could not be obtained.

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* Chief Engineer, Metals and Nonmetals Division, U. S. Bureau of Mines.

Construction and Equipment of the Ross Shaft, Homestake Mining Company

By Guy N. Bjorge* and A. J. M. Ross,† Members A.I.M.E., J. D. Johnson,† Member,‡ S. J. Staple† and J. F. Wiggert,† Member§

(New York Meeting, February, 1935)

In recent years the Homestake mine has been served by three shafts, the B. &. M., the B. & M. No. 2 and the Ellison, supplemented by an inside shaft, the Milliken, extending from the 2000-ft. level to the 3200-ft. level. In 1932, when a new operating shaft was under consideration, the bottom of the mine was at the 3200-ft. level. The B. & M. and the B. & M. No. 2 shafts were down to the 1550-ft. level and the Ellison was being deepened from the 2600 to the 3200-ft. level.

All ore was hoisted at the B. & M. and Ellison shafts. Levels below the 2600 level, which were under development only, were served by the Milliken shaft. Below the 1550-ft. level, the mine was served only by the Ellison shaft from the surface. Within a few years, when ore reserves above the 1550-ft. level would be depleted, operation would be limited to the Ellison shaft and the auxiliary underground shaft. The Ellison shaft alone was not adequate for hoisting the entire output from the deeper levels and also handling all men and supplies. Furthermore, assurance of safety and uninterrupted operation required two fully equipped shafts to the deeper levels. Therefore a new operating shaft was authorized in November, 1932, and was named for A. J. M. Ross, the mine superintendent.

Plans for New Shaft

The B. & M. shaft, begun in 1878 before much was known of the form and extent of the orebodies, was too near the ore and was endangered by ground movement. In 1924 this shaft was seriously threatened and sinking of the B. & M. No. 2 shaft, to supplant it, was begun. The latter was completed to the 1550-ft. level in 1928. In the meantime ground movement at the B. & M. shaft had practically stopped and, though loss of the shaft constantly threatened, ore hoisting was con-

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* Assistant General Manager, Homestake Mining Co., Lead, S. D.

† Mining Superintendent, Chief Engineer, Master Mechanic, Chief Electrical Engineer, respectively, Homestake Mining Co.

‡ Since Nov. 10, 1037.

§ Since April 16, 1937.

tinued until December, 1934, when the Ross shaft was placed in operation. Except for the emergency period following the fire, which destroyed the Ellison hoist buildings in July, 1930, the B. & M. No. 2 shaft has been used only for lowering timber.

Sinking and further utilization of the B. & M. No. 2 shaft for deeper levels could not be considered because of its remoteness from the orebodies at greater depth. Furthermore, it was estimated that the cost of enlarging this shaft for greater capacity, sinking and equipping it for the ultimate depth contemplated, with the necessary underground connections, would exceed the cost of a new shaft, which might be located more advantageously.

Selection of Site for Ross Shaft.—The combination of surface topography, the location of mine plant and mills and the pitch of the Homestake lode placed rather close restrictions on the sites that might be utilized for a new shaft. In selecting the site, it was necessary to consider its relation to existing surface plants, to underground operation on existing levels, and to the development of deeper levels. The new shaft must be south of the Ellison shaft and at an elevation that would allow for the primary crushing plant and an adequate storage bin between the skip dump and the level of the tramway leading to the South mill, where the total output would eventually go for treatment. Because of the very steep slopes into the canyon of Whitewood Creek, the only suitable site was at the top of a long narrow ridge approximately 2000 ft. south of the Ellison shaft (Fig. 1). This could be made accessible by road and railroad for the delivery of material, machinery and equipment, and of all possible sites it was most accessible to the South mill and shops. Also, it could be reached by a relatively short adit from Whitewood Gulch, at the elevation of the 300-ft. level, and by short crosscuts from existing drifts on the 800-ft. level and the 1400-ft. level, for raising the upper part of the shaft.

Ultimate Depth.—The ultimate depth for which the shaft should be equipped was the next consideration. It was, of course, necessary to provide for further deepening of the mine. The 3200-ft. level was then the lowest and deepening to the 3500-ft. level was in progress. The depth of the 5000-ft. level was finally selected as the maximum from which hoisting in one vertical lift would be feasible and efficient. The elevation at the collar of the new shaft was 120 ft. above the datum for mine levels. The ultimate depth from the collar to a skip-loading pocket below the 5000-ft. level would then be 5200 feet.

Fireproof Construction.—Because of experience in the fire that destroyed the Ellison hoist buildings, the first specification laid down was that the entire surface plant and the shaft to a depth of at least 500 ft. below the collar must be of fireproof construction. As plans progressed this specification was extended to include the full depth of the shaft, all

shaft stations and skip-loading pockets. This specification has been closely followed. The only timber used in the shaft or adjacent to the shaft on mine levels is for guides and blocking.

Arrangement of Ross Shaft

The shaft has six main compartments, as follows: one cage compartment 13 ft. by 6 ft. 1 in.; two skip compartments each 5 ft. 7 in. by 5 ft.

Fig. 1.—Plan showing location of Ross shaft in relation to existing mine plant and mills.

8 in.; one pipe compartment 6 ft. 10 in. by 3 ft. 7 in.; one ladder compartment 6 ft. 10 in. by 3 ft. 6 in.; and one counterweight compartment 6 ft. 10 in. by 3 ft. 7 in. The counterweight compartment has one subdivision for the counterweight, one for electric cables and one for sollars on which men can work. See Figs. 2 and 3.

The controlling factor in the determination of the size and arrangement of the shaft was the desire to have a cage long enough to take all

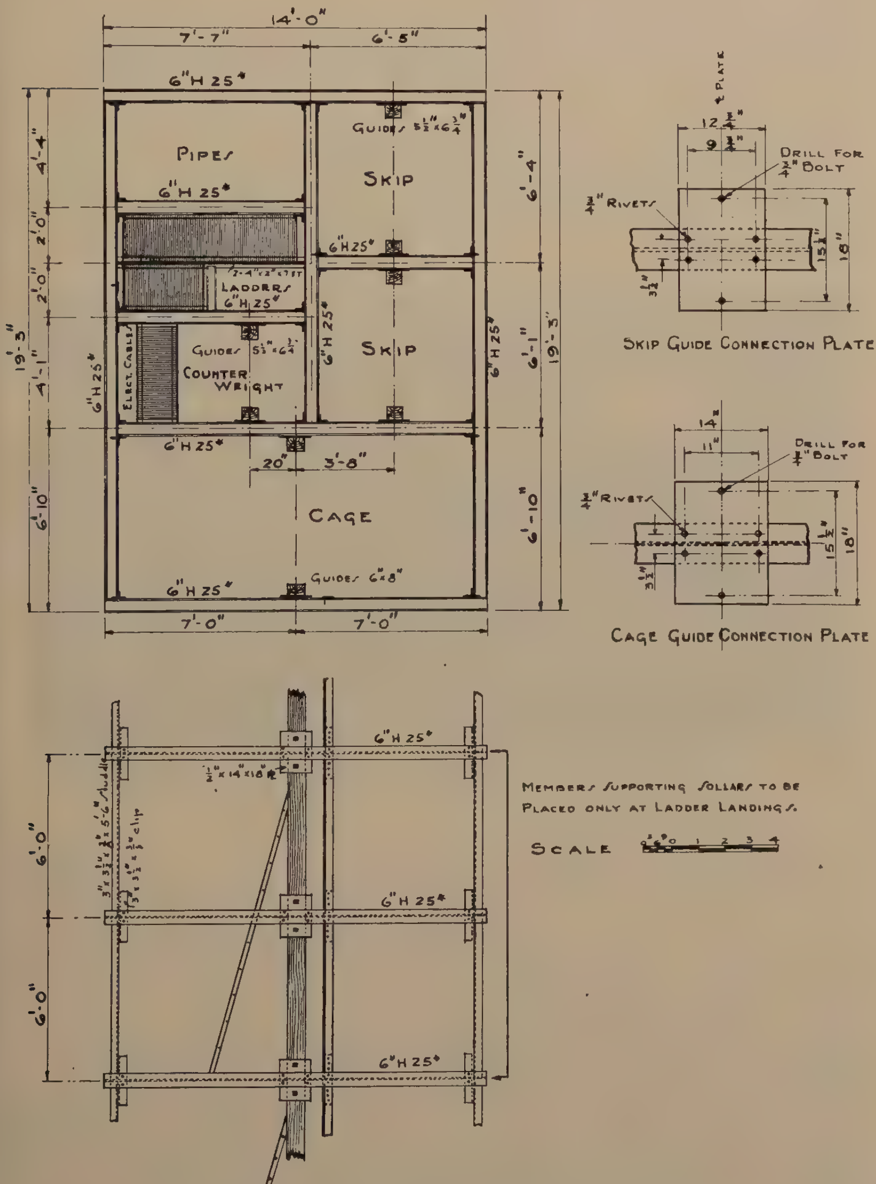


FIG. 2.—STANDARD STEEL SET FOR ROSS SHAFT.

possible material into the mine in truck loads without rehandling. To fulfill this requirement the cage must take 12-ft. material. The cage compartment, therefore, was made 13 ft. long with the cage 12 ft. 7 in.

long. The skip compartments were planned to accommodate the 7-ton skips in use at the Ellison shaft. The outside measurements of the shaft sets are 19 ft. 3 in. by 14 ft.

To fulfill the requirement of fireproof construction, steel shaft sets and concrete were given consideration. A material cost differential in favor of steel sets was estimated, therefore steel sets were selected. This decision could not have been reached if the condition of the ground had required the additional bearing provided by concrete. There is practically no ground in the Homestake mine that does not stand indefinitely without support in ordinary mine openings, therefore steel sets are adequate.

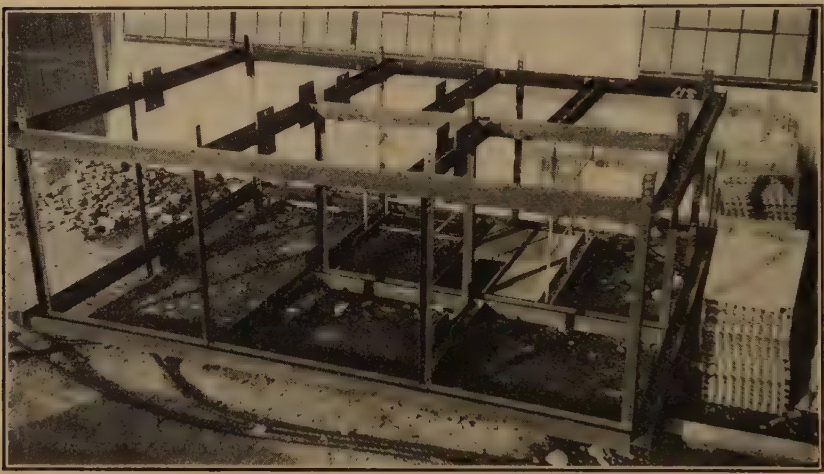


FIG. 3.—ASSEMBLY OF STEEL SETS.

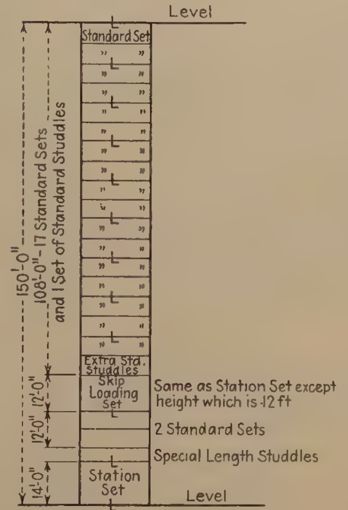
Steel shaft sets were designed by the Homestake staff after consultation with the engineers of the steel fabricators, and finally checked after inspection of the Godfrey shaft at Hibbing, Minn. and the Newport and Geneva shafts at Ironwood, Mich. All wall plates, end plates and dividers are 6-in. 25-lb. H-beams. Studdles are 3½ by 3 by ⅜-in. angles. At all shaft stations and skip-loading pockets 6-in. H-columns take the place of studdles through two sets on the open side of the shaft. Bearing beams are 12-in. 50-lb. I-beams, 20 ft. long except in a few places where the ground made longer bearing beams necessary. Special beams are up to 26 ft. long. Details of shafts are shown in Fig. 2 and the schedule of steel for the shaft in Fig. 4. The specification for all steel for the shaft is that it shall contain between 0.20 and 0.25 per cent copper. The steel was furnished by the American Bridge Company.

Shaft bracing consists of 12 by 12-in. Oregon fir. All wedges are also of Oregon fir. Guides are of native pine bolted to plates, which are riveted to end plates and dividers. In the skip compartments guides are

5½ by 6¾ in. and the plates to which they are attached are 12¾ by 18 by ½ in. The cage-compartment guides are 6 by 8 in. and the guide plates 14 by 18 by ½ inch.

The skip compartments are laced with 14-gage galvanized corrugated copper-bearing steel held in place by bolted clips measuring 4 by 2 by ⅛ inch.

Level	Elev.	Standard Sets	Special Length Studdles	Station Sets	Skip Loading Sets	Extra Sets of Standard Studdles	Distance between Levels
Collar of Shaft	5355.00						
Tramway Level	5193.65	25	1 Set	1			161.35'
300-ft. Level	4930.00	41	"	1			263.65'
800-ft. Level	4430.85	80	"	1			499.15'
1400-ft. Level	3829.34	97	"	1			601.51'
1700-ft. Level	3529.00	47	"	1			300.34'
1850-ft. Level	3379.00	22	"	1			150'
2000-ft. Level	3229.00	22	"	1			150'
2150-ft. Level	3079.00	19	"	1	1	1	150'
2300-ft. Level	2929.00	22	"	1			150'
2450-ft. Level	2779.00	22	"	1			150'
2600-ft. Level	2629.00	22	"	1			150'
2750-ft. Level	2479.00	19	"	1	1	1	150'
2900-ft. Level	2329.00	22	"	1			150'
3050-ft Level	2179.00	22	"	1			150'
3200-ft Level	2029.00	22	"	1			150'
3350-ft. Level	1879.00	19	"	1	1	1	150'
3500-ft. Level	1729.00	22	"	1			150'
3650-ft. Level	1579.00	22	"	1			150'
3800-ft. Level	1429.00	22	"	1			150'
3950-ft. Level	1279.00	19	"	1	1	1	150'
Totals		608	20	20	4	4	



NOTE.

Standard Set consists of One Frame and Studdles to next Frame above

Station Set and Skip-loading Sets consists of 3 Frames and 2 Sets of connecting Studdles as shown

Ladder Landings to be placed as indicated in above sketch by letter "L"

FIG. 4.—STEEL SCHEDULE FOR ROSS SHAFT.

All shaft ladders are of steel. The rungs are ¾ in. round and the risers 2 by ⅜ in. Standard ladders are 14 ft. 6 in. long and 12 in. wide. All ladders are staggered (Fig. 2).

All sollars are steel gratings made of 1 by $\frac{3}{16}$-in. steel bars held in place by $\frac{7}{16}$-in. steel rods. The bars are spaced ¾ in. apart by ¾-in. ring spacers. The arrangement of sollar gratings is shown in Fig. 5.

A space 8 in. by 3 ft. 7 in. in the counterweight compartment is provided for electric cables. Details of these cables and their support are shown in Fig. 6.

All pipes entering the mine through this shaft are supported on 12-in. I-beams. These beams are set in hitches cut in the shaft walls, which

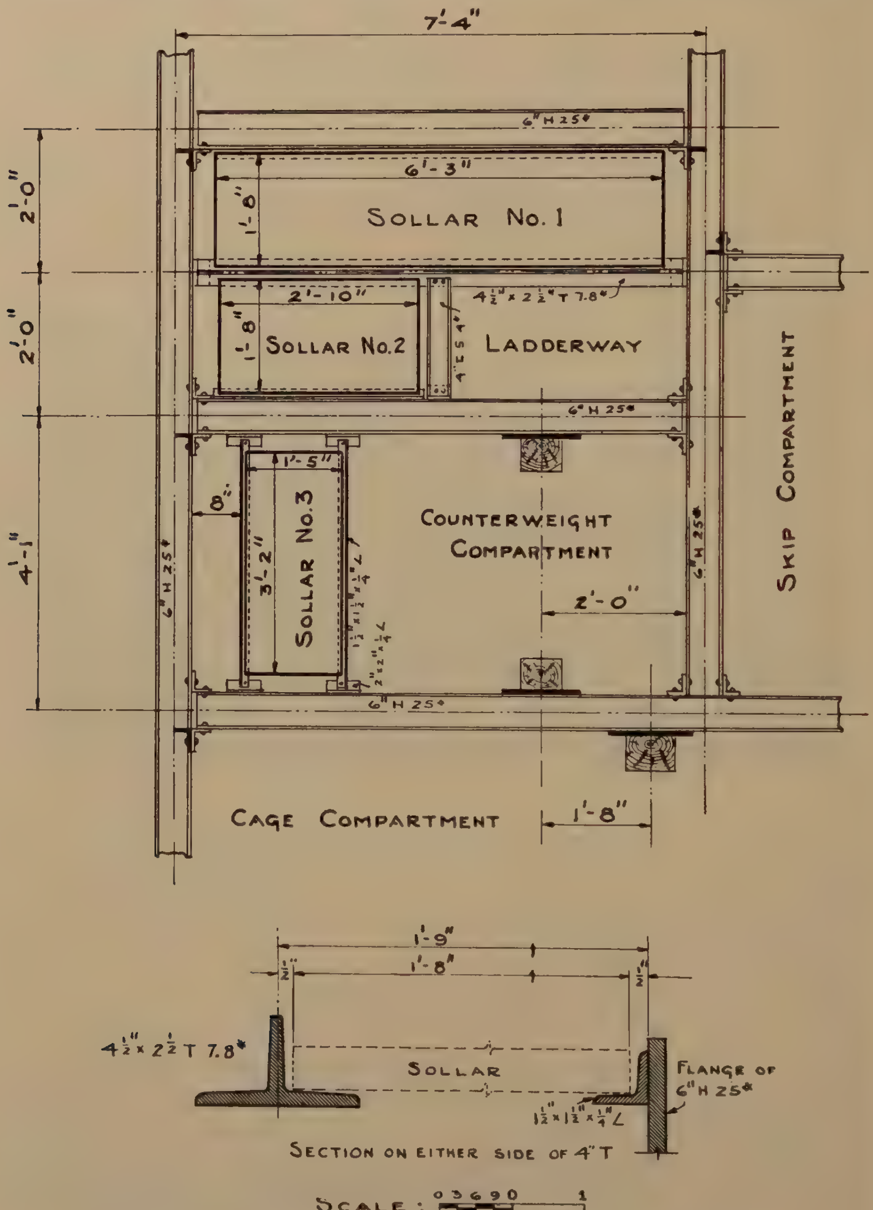


Fig. 5.—Plan showing position of sollars and their supports.

are then filled with concrete. The outer end of the beams are supported by knee braces of 6 in. by 4 in. by ½-in. angles set in hitches filled with

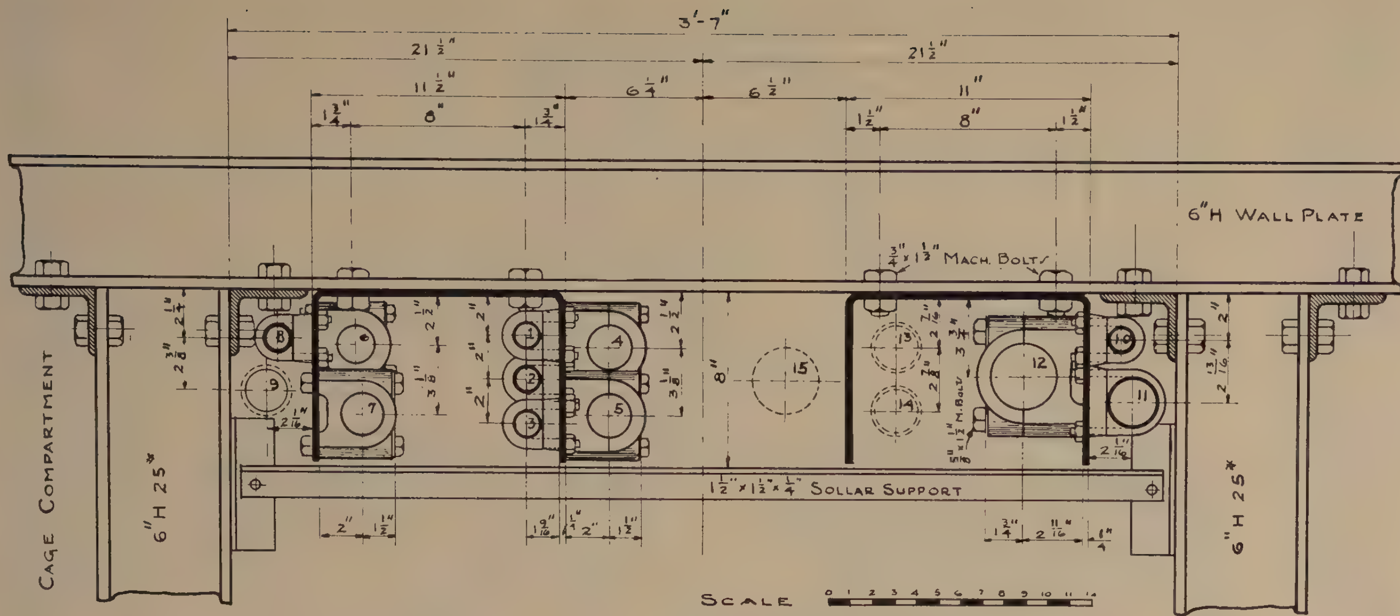


FIG. 6.—PLAN OF SHAFT CABLES AND CONDUITS IN ROSS SHAFT, SHOWING METHOD OF SUPPORT.

1. Man-hoist cage and counterweight. 1-in. dia. conduit ⎫ Secondary
2. Ore-hoist south skip. 1-in. dia. conduit ⎬ signal
3. Ore-hoist north skip. 1-in. dia. conduit ⎭ conduits.

4 and 5. Signal cables.

6. Telephone cable.
7. Signal cable.
8. Telephone. 1-in. conduit, secondary phone
9. Spare 2-in. conduit space.
10. Call horns, 1-in. dia. conduit.
11. Lighting conduit, 2 in. dia.
12. Present power cable, three 350,000 cu. m. wires.
13. Spare conduit space, 2-in. dia.
14. Spare conduit space, 2-in. dia.
15. Position of second power cable if installed.

concrete. Details are shown in Fig. 7. All pipes are conducted from the shaft through 6 ft. by 6 ft. drifts cut into the shaft on the manway side. See Fig. 8. This leaves the station proper free of pipes except those for drinking water.

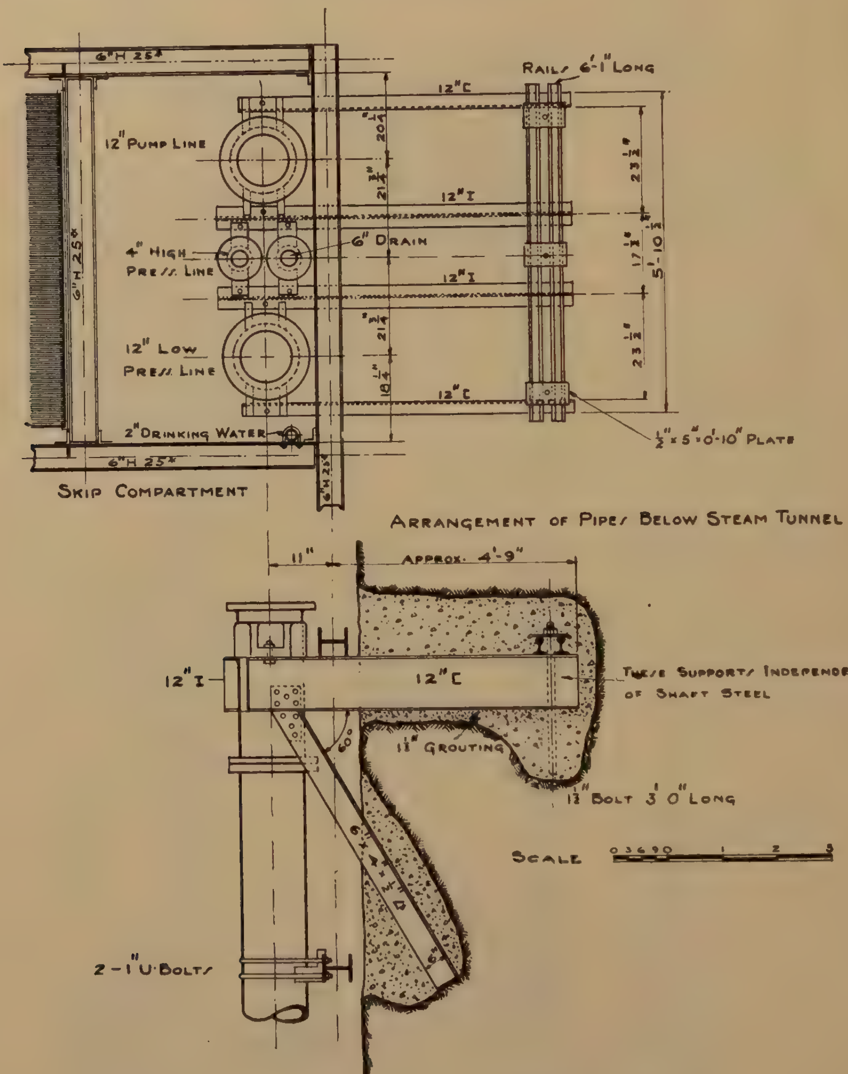


Fig. 7.—Details of pipe supports in shaft.

Shaft Stations and Ore Passes

Shaft stations are standardized on all levels except for the angle at which the drifts pass the shaft. All levels below and including the

1250 level are 150 ft. apart and 7 by 7-ft. drifts are standard throughout nearly all of the mine. The center of the main haulage drift is 35 ft. in front of the shaft. For 100 ft. in each direction from the shaft station drifts are widened to 9 ft. to provide for double tracking. In the direction

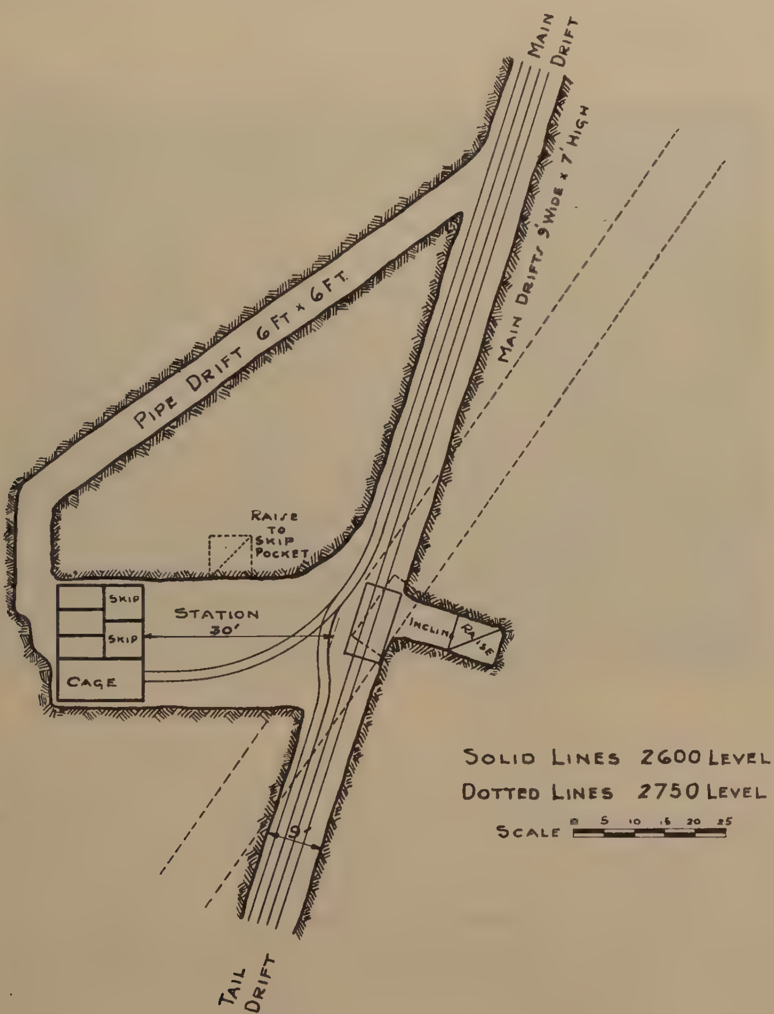


FIG. 8.—GENERAL PLAN OF SHAFT STATIONS.

away from the orebodies the drift is reduced to the standard 7 by 7-ft. drift and continued another 100 ft. for tail room (Fig. 8).

The station in front of the shaft is cut from drift height, or 7 ft. at the haulage drift to a height of 13 ft. 6 in. in the clear at the front line of the shaft. Outward from the shaft three steel station sets are spaced

at 6-ft. centers. These sets are roofed with 14-gage galvanized corrugated steel. The remainder of the station has no support. See Fig. 9.

Skip-loading stations are planned at 600-ft. intervals, the first one being 40 ft. above the 2150-ft. level. From levels between these loading stations ore is passed downward through 6 by 6-ft. untimbered raises. Air-operated gates are installed in these ore passes on each level so that the flow of ore may be controlled.

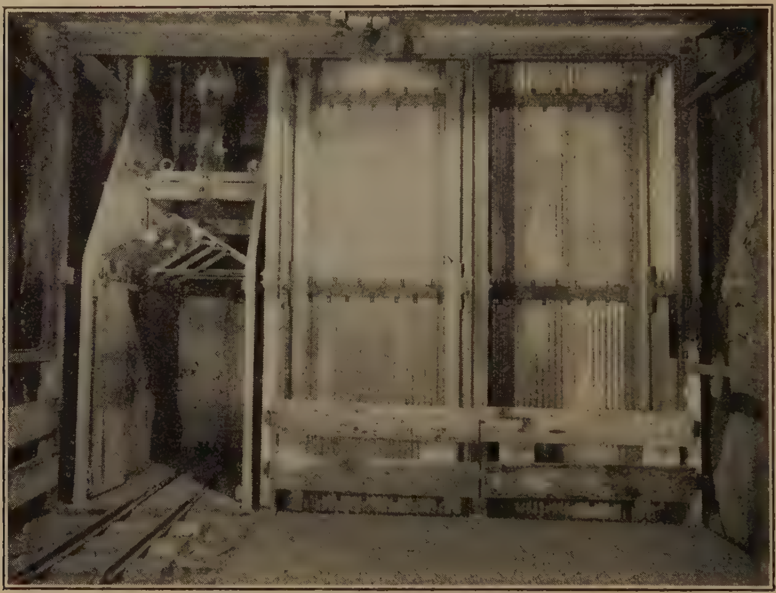


Fig. 9.—Typical shaft station.

Two loading stations, one at 40 ft. above the 2150 level and one at 40 ft. above the 2750 level, are now completed and in operation; 6 by 6-ft. raises were put up from the level immediately below to each loading station. These served for drawing off rock while mining was being done. On completion of construction they are equipped as manways, as the loading stations are not accessible to the ladderway in the shaft. Within four months a third loading station, at 40 ft. above the 3350 level, will be completed. The general arrangement of ore passes and skip-loading pockets is shown in Fig. 10.

No steel or timber is necessary for ground support in the skip-loading stations. Concrete and steel are used to support chute gates and skip-loading equipment. The skip loaders are almost identical with those that have long been in use at our other shafts. They are of the double gate type. The measuring and loading mechanism sets, of which there are two at each station, one for each skip, are installed in a room or pocket adjacent to the shaft and below the ore raise or bin. This room

is entered from the level below, or from the shaft; it is approximately 37 ft. high, 22 ft. wide and 15 ft. deep.

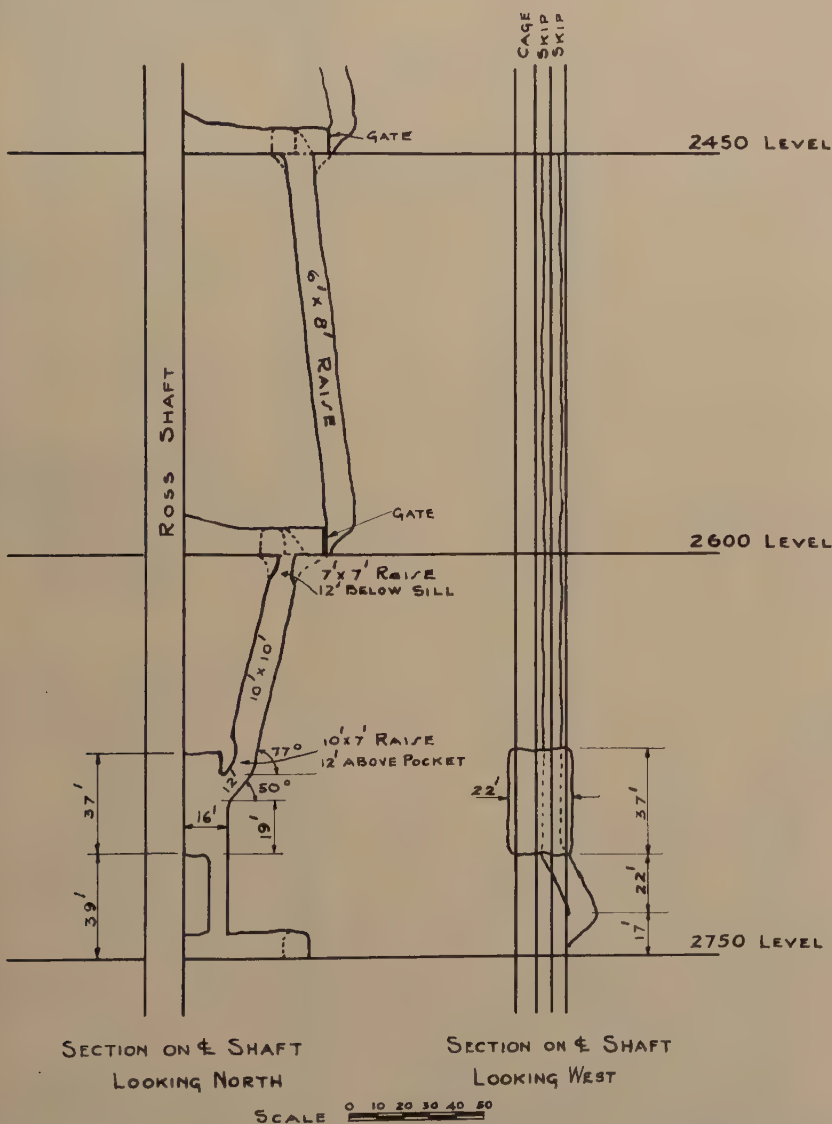


FIG. 10.—SECTIONS SHOWING GENERAL ARRANGEMENT OF SKIP POCKET AND ORE RAISES.

The equipment consists of a hand-operated rack and pinion gate installed at the ore raise or bin outlet. The skip-loading mechanism is composed of a steel chute placed at a slope of 45°, extending from the ore raise gate to the shaft, a vertically rising gate 5½ ft. from the ore-

raise gate, and an outward-swinging gate 10 ft. from the vertical gate. The capacity of the chute between these two gates is 145 cu. ft., or one skip load. These gates, through a series of steel levers and rods operated by a vertical air cylinder $15^{19}/_{32}$ in. in diameter, using compressed air at 100 lb., are arranged so that the vertical gate is raised through the ore stream cutting off the supply from the ore raise and continuing up with the levers sufficiently to then open the swinging gate and discharge the ore into the skip. The normal loading period is six seconds. Reversal of the mechanism progressively closes the swinging gate and lowers the vertical gate, allowing the ore stream to again fill the measuring compartment. The hand-operated ore-raise gate is left open while skipping ore, except in emergencies. The chutes have bottom liners of heavy cast iron and side liners of steel plate. The gates are lined with steel plate.

The operator's platform, with control levers, is between the loaders, 16 ft. above the chute discharge and near the ore-raise gate, from which position observation of the skip loading is made through an iron grating in the platform and any "hang-up" of the ore in the raise is conveniently cleared. A hand-operated 10-ton traveler is installed near the ceiling of the pockets to facilitate the handling of loader parts during installation and the making of repairs. The excavations for skip-loading stations is shown in Fig. 11 and the general assembly of the skip loaders in Fig. 12.

Shaft Construction

The plan followed in shaft construction was to drift or crosscut to the site of the shaft from existing mine workings on various levels, put up pilot raises between these levels, then strip to full shaft size and install steel.

Drifts.—Originally drifts were driven to the shaft site on the 300 level, 800 level, 1400 level, 1700 level, 2000 level and all levels below. The shortest of these drifts was 320 ft. and the longest was 1700 ft. These levels served for shaft construction. Later drifts to the shaft to serve operating purposes were also started on the 1250 level, 1550 level and 1850 level. Drifts on these levels have not yet reached the shaft. All drifts to the shaft are standard 7 by 7 ft. and untimbered. All drifting was by miners working on a contract basis at a fixed rate per foot. The contract rate included powder, fuse and caps; all other material was furnished by the company. Until Jan. 1, 1935, total drifting, including the 7 by 9-ft. sections at the shaft on each level, was 19,413 ft. The average cost per foot is given in Table 1. The tramway adit, timbered with 10 by 10-in. native pine, is 2034.5 ft. long. Its dimensions inside the timbers are 7 ft. 4 in. wide at the cap, 9 ft. wide at the sill and 7 ft. 10 in. high.

Pilot Raises.—Except in 137 ft. which was sunk from the surface and 250 ft. which was raised full shaft size from the 800-ft. level, pilot

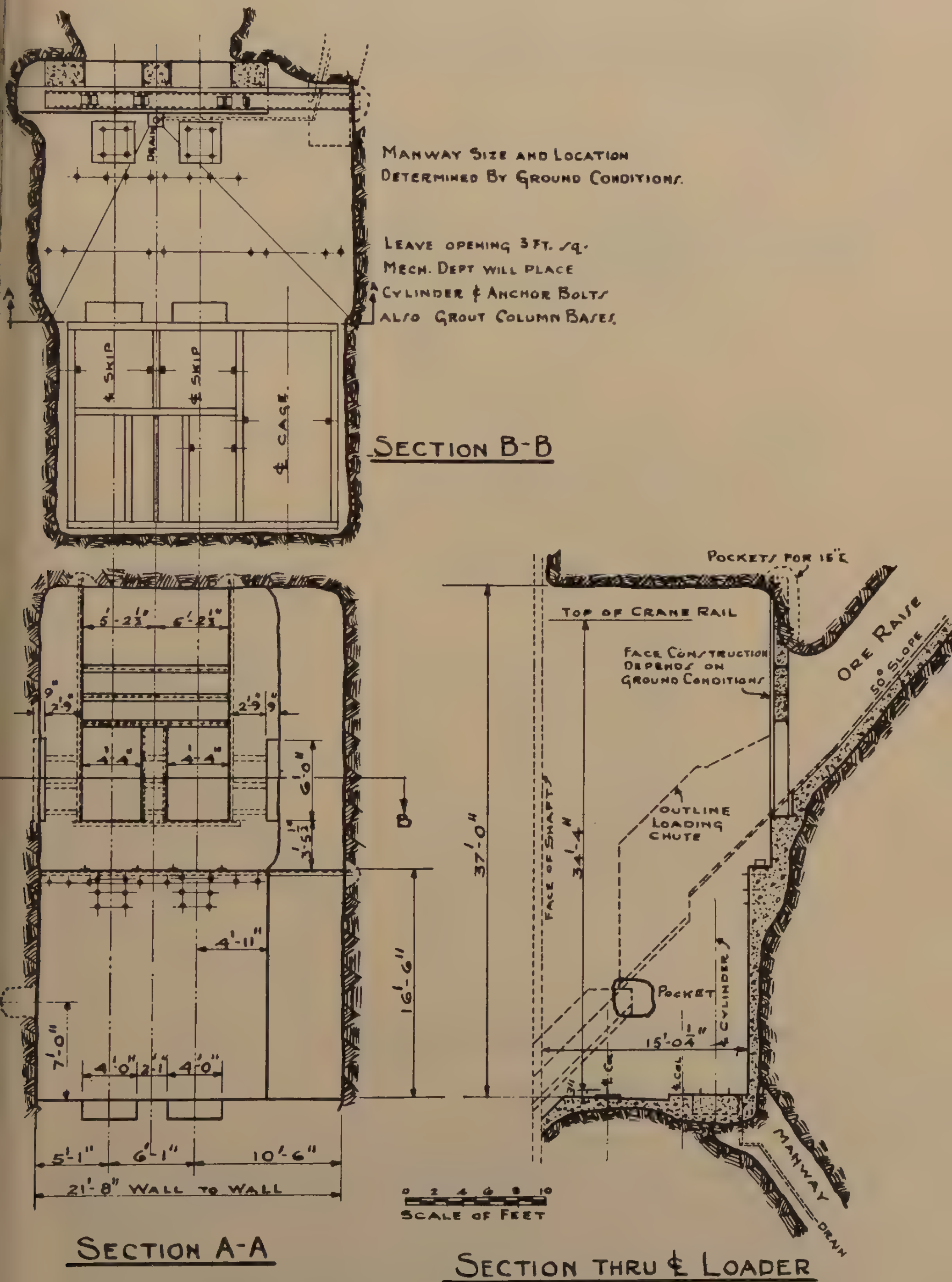


FIG. 11.—EXCAVATION FOR SKIP-LOADING POCKET.

raises were put up from level to level throughout the depth of the shaft. These raises were 6 by 6 ft., in the center of the full shaft area. The height of each raise was as shown on p. 73.

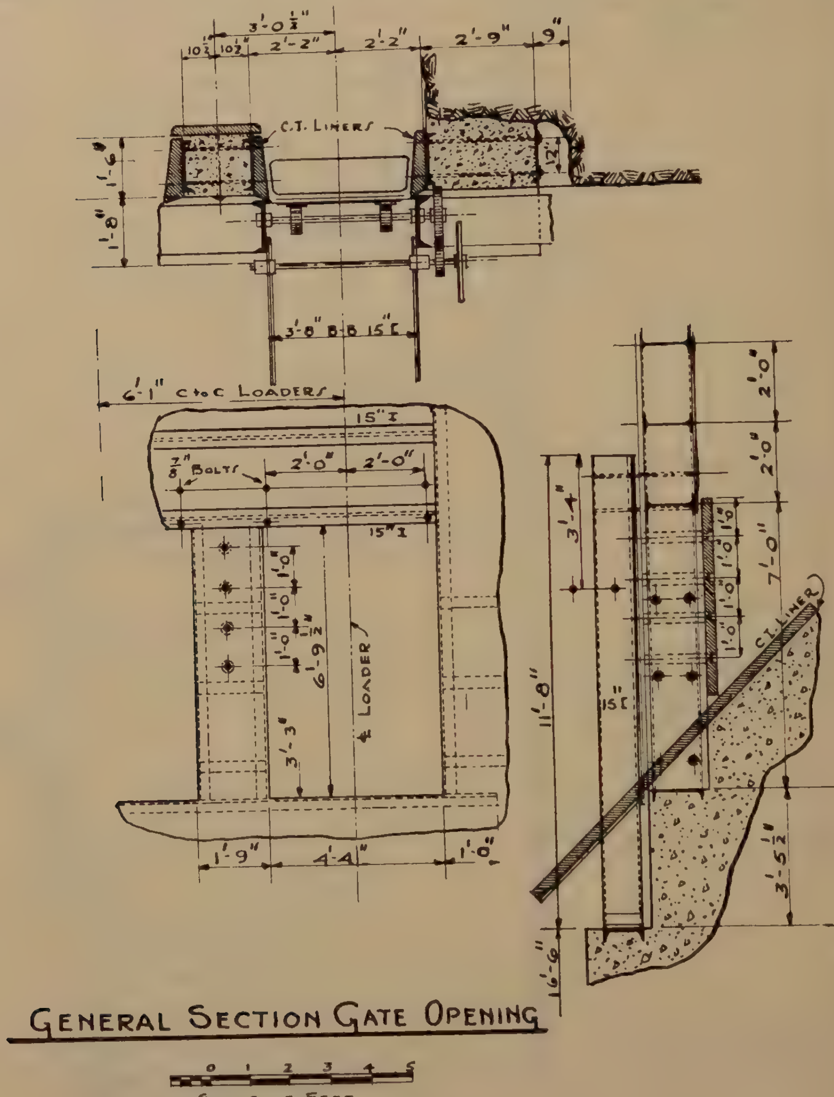


FIG. 11.—(*Continued.*)

Raising full shaft size was tried in the raise from the 800-ft. level. When it reached a height of 250 ft. the idea was abandoned because crushing of timbers under the weight of the column of broken rock, which was continually moving, was rapidly approaching the limit for safe working. Therefore 6 by 6-ft. pilot raise was continued to the 300 level. The

	Feet
Adit (300 level) to shaft sunk from surface	276
800 level to adit	500
1400 level to 800 level	600
1700 level to 1400 level	300
2000 level to 1700 level	300
All raises between 2000 and 3200 levels	150
3500 level to 3200 level	300

total footage of pilot raises, including the shaft-size section and the sinking from the surface, up to Jan. 1, 1935, was 3392 ft. The work was done on a contract basis. The cost is shown in Table 1:

TABLE 1.—*Some Costs of Shaft Construction*
UP TO JAN. 1, 1935

	Drifts, Average Cost per Foot	Pilot Raises, Cost per Foot	Stripping, Cost per Foot of Shaft
Mining:			
Labor	$ 5.52	$ 9.19	$13.77
Explosives	2.15	2.73	3.11
Air and air drills	1.39	3.20	2.54
Timbering	0.20	4.24	0.67
Track:			
Labor	0.40	0.34[a]	0.56[a]
Material	1.10		
Pipe:			
Labor	0.07	0.49	0.25
Material	0.29		
Hoist installation and operation		1.82	1.26
Haulage:			
Labor	0.22	0.85	2.87
Power	0.23		
Electrical supplies			0.05
Miscellaneous:			
Labor	0.30	1.13	1.43
Material	0.10		
Surveying	0.09	0.12	
Total	$12.06	$24.11	$26.51

[a] Temporary loading tracks.

Stripping and Steel Installation.—During a large part of the time stripping and installation of steel were in progress in three parts of the shaft at the same time. The work was carried on in three eight-hour shifts. Each crew consisted of one shift-boss and four shaft men on each shift. In all, three shift bosses and 36 shaft men were engaged in this work. All men were from the regular underground force and under the supervision of the regular foremen and superintendents. No addi-

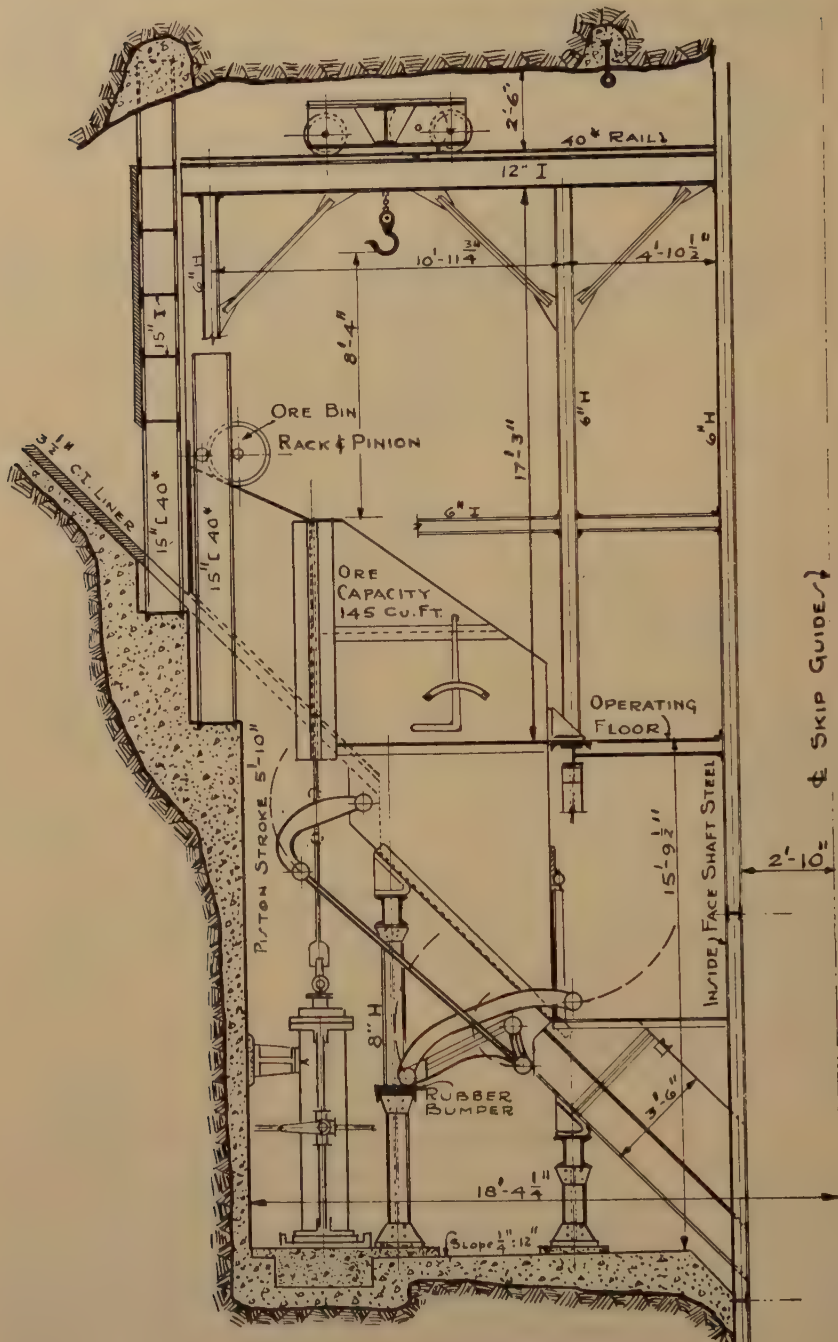


FIG. 12.—GENERAL ASSEMBLY OF SKIP LOADERS.

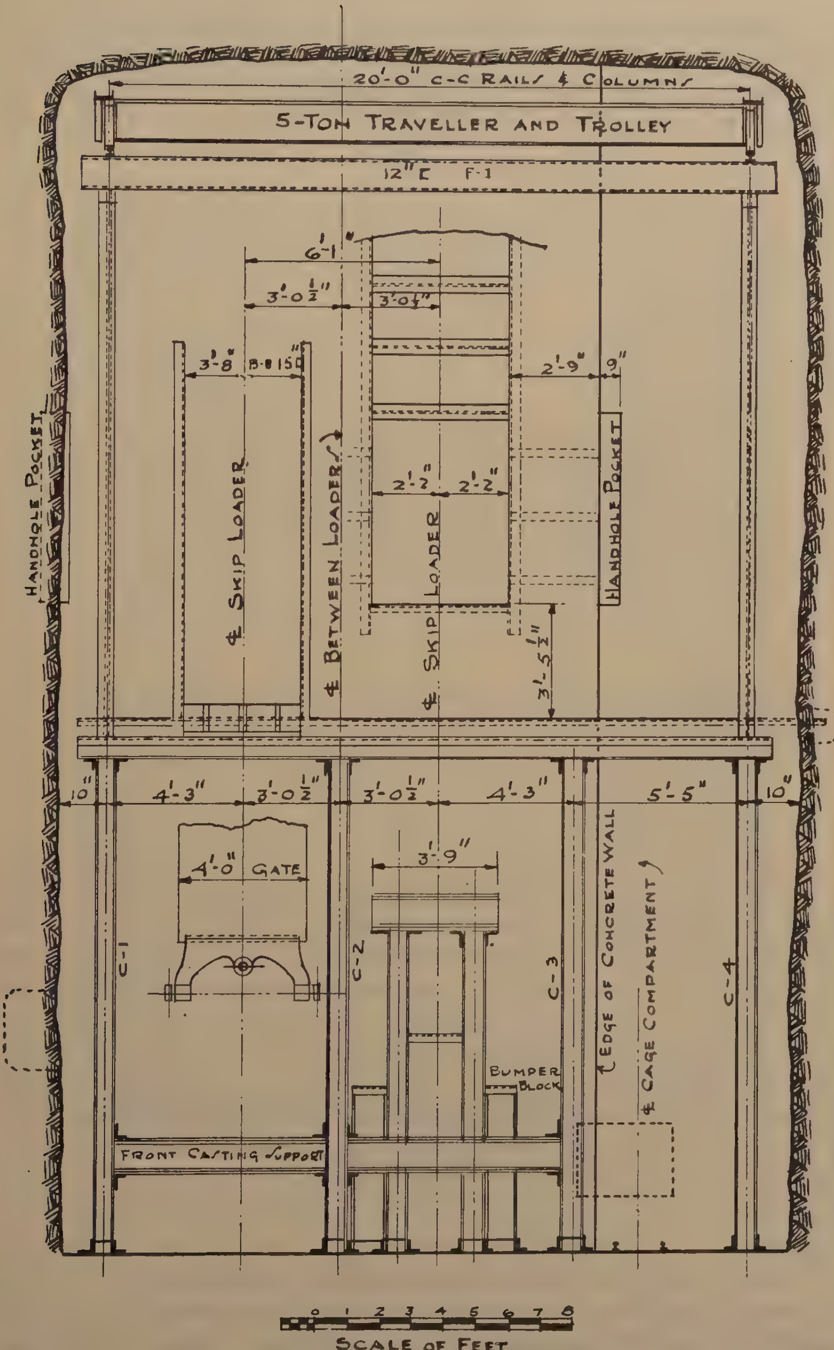


Fig. 12.—(*Continued.*)

tions to the regular mine staff were made for the shaft work. The cost of all stripping per foot of shaft to Jan. 1, 1935 is given in Table 1. Up to Jan. 1, 1935, steel sets had been installed for a total of 3241.5 ft. of the shaft. The cost per foot is given in Table 2.

TABLE 2.—*Cost of Steel Sets, Ross Shaft*

	Cost per Foot	
Steel: invoice cost	$21.13	
Unloading and handling	2.08	$23.21
Installing steel:		
Labor	6.92	
Power	0.33	
Labor on hoists	2.15	
Air and air drills	0.12	
Supplies for hoists	0.11	
Miscellaneous supplies	3.10	12.73
Corrugated lacing:		
Invoice cost	6.35	
Clips for fastening	0.39	
Installing labor	1.06	
Labor on hoists	0.11	
Miscellaneous material	0.13	8.04
Sollar gratings		0.97
Ladders		0.68
Railings on manway landings		0.87
Shaft doors		0.17
Guides and chairs		3.33
Surveying		0.32
Total		$50.32

Shaft Concreting and Gunite.—From the surface to a depth of 308 ft. the shaft is concreted solidly outside of the steel sets. Details of forms are shown in Fig. 13. The concrete was mixed on the surface and passed down the shaft through a 4-in. wrought-iron pipe, from which it discharged into a hopper hanging from four rope blocks. Four chutes led from the discharge hopper to the walls of the shaft. A platform staging hung from rope blocks in the cage compartment was raised as required for form placing. Forms were moved on day shift and concrete poured on night shift. One set, or 6 ft., was poured per day; 1060 cu. yd. of concrete was used, at a cost of $13.46 per cu. yd., or $46.27 per foot of shaft.

Below 308 ft., the shaft was concreted in the same way for one, two or three sets wherever there was the slightest question as to the safety of the ground. The sections needing concrete totaled 150 ft. The concrete for this work was mixed on the level above where it was needed, in a small compressed-air concrete mixer, and passed down through an 8-in. pipe.

All of the shaft that is not lined with concrete is given a gunite coating. This work is now complete to the 2000-ft. level. The guniting crew consisted of four men on each shift with one boss. More time was required in washing the shaft walls than in the actual placing of

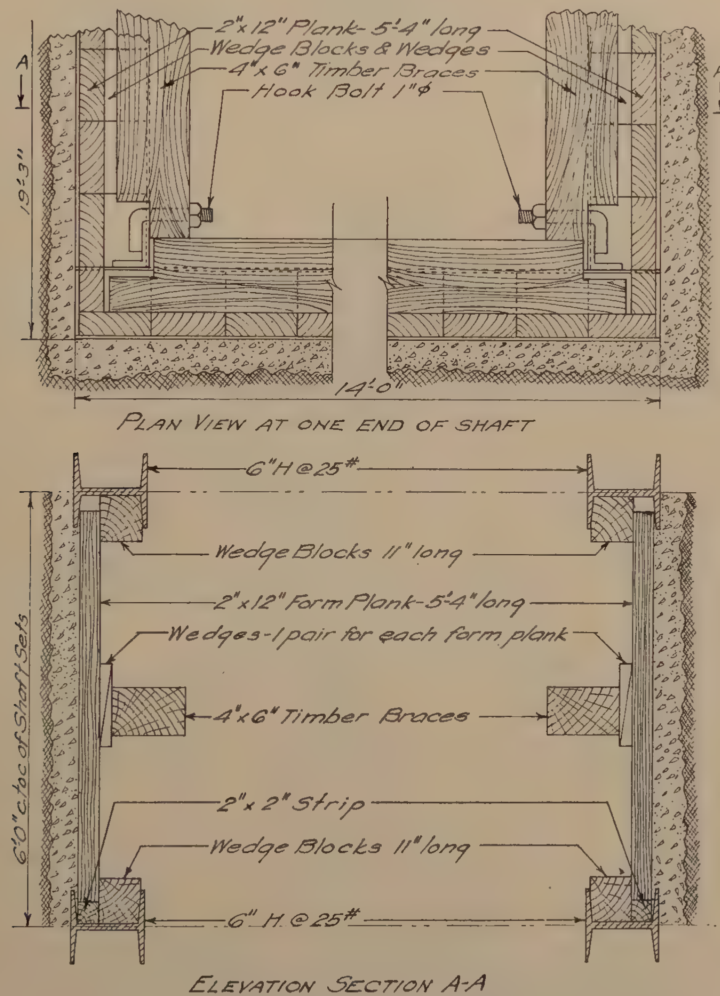
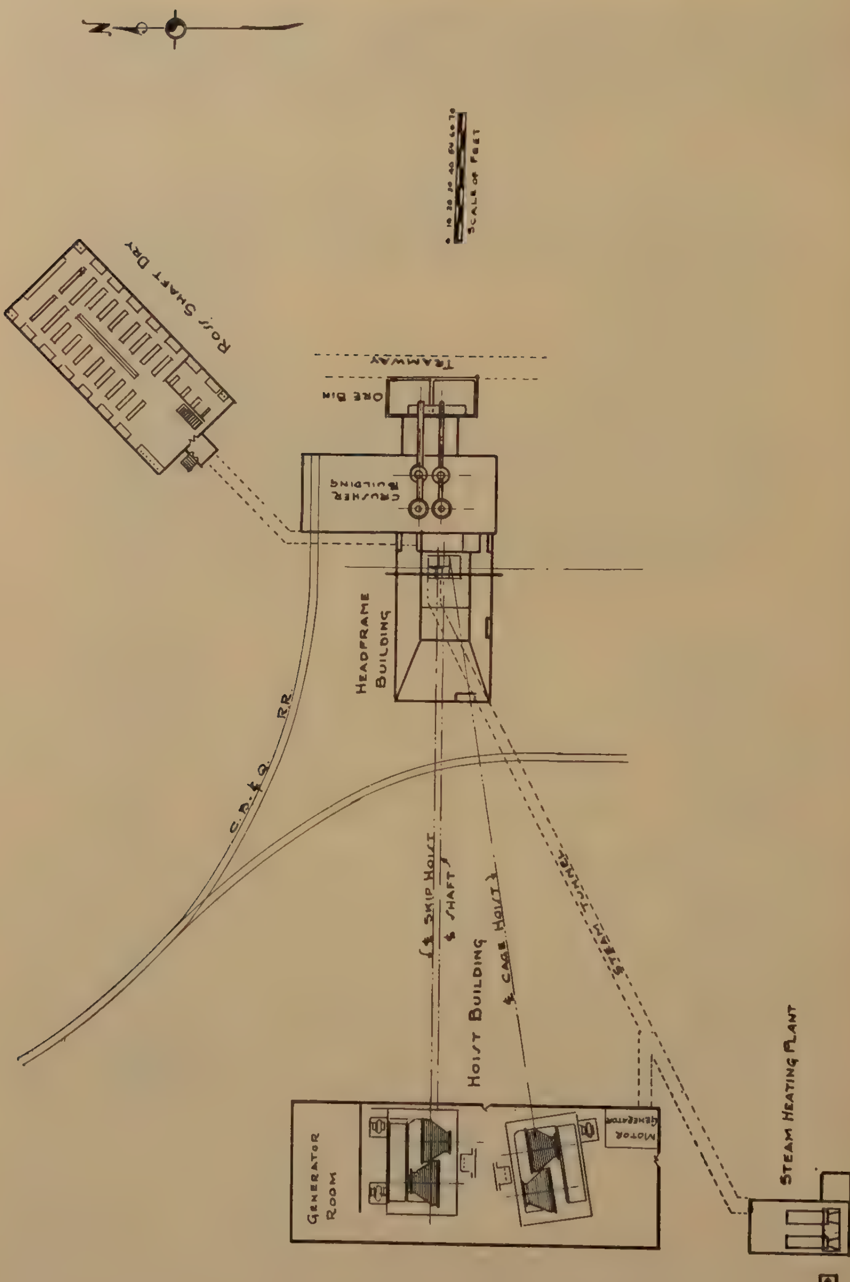


FIG. 13.—DETAILS OF FORMS FOR CONCRETE LINING IN SHAFT.

gunite. A large part of the shaft walls had to be gone over with wire brushes, as water alone would not remove the coating of mud. The gunite will greatly speed the work of shaft inspection, as the slightest cracking or movement in the rock will immediately be evident in the gunite coating.


N
Ross Shaft Dry
Tramway
Ore Bin
Crusher Building
Headframe Building
C.B.&Q. RR.
℄ Skip Hoist
℄ Shaft
℄ Cage Hoist
Steam Tunnel
Hoist Building
Generator Room
Motor Generator
Steam Heating Plant
Scale of Feet

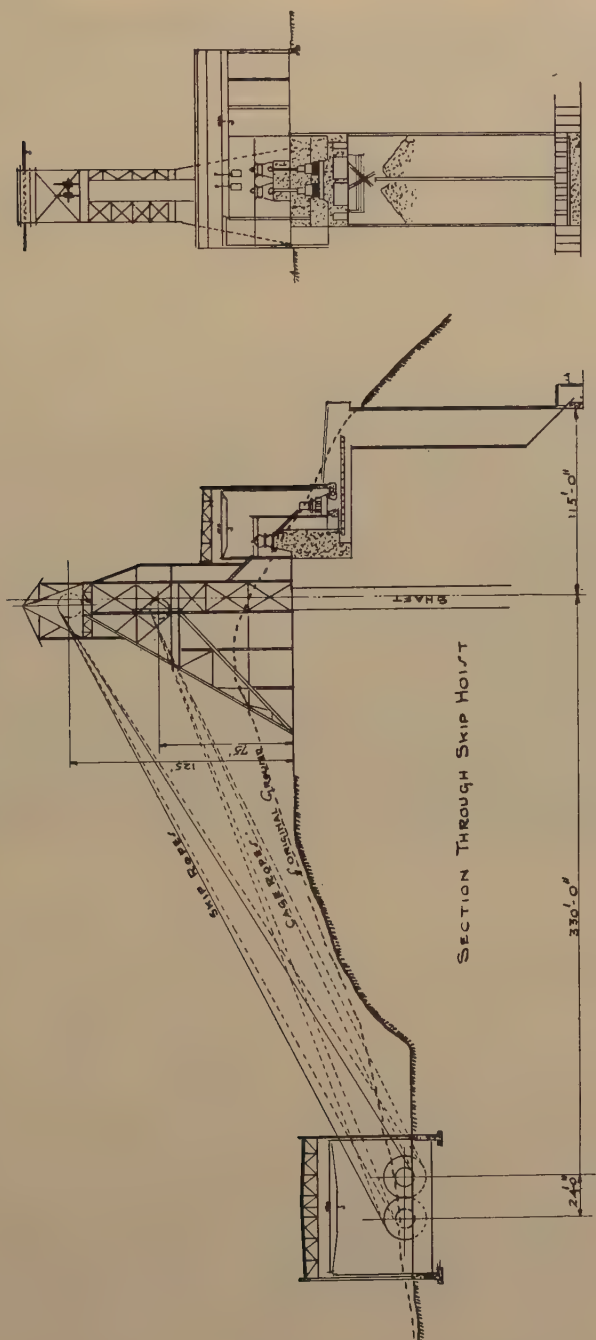


Fig. 14.—Surface plant in plan and elevation, Ross shaft.

A summary of shaft costs per foot for the 3241.5 ft. in which steel was installed, as of Jan. 1, 1935, follows:

Pilot raising	$ 24.11
Stripping	26.51
Installing steel	50.32
Concreting to depth of 308 ft	4.43
Concrete and gunite to 2450 level	6.97
Stations	5.49
Pipe lines in shaft	0.53
Signal and electric lines	13.29
General construction	0.66
Total	$132.31

Surveying.—Thirteen pilot raises ranging in height from 150 to 600 ft. have been completed. All holed through at the proper location. In this part of the job some slight leeway was possible because the 6 by 6-ft. pilot raises were in the center of the 15 by 20-ft. (in the rock) shaft area.

The shaft was then plumbed through the pilot raises to set points for steel installation. From the surface to the 300-ft. level the maximum possible distance between wires was 2.3 ft. At all other places it was 4 ft. Installation of steel sets began at the surface, and at the 300, the 1400, the 2150 and the 3050 levels. All connections have been made. The largest variation from true vertical alignment at any corner of the shaft at any of the connections is ⅝ in. This was at one place only. For all other points the maximum was ⅜ inch.

Progress.—The most important drifts to the shaft site were started immediately after the shaft was authorized on Nov. 30, 1932. Progress

TABLE 3.—*Progress of Work on Ross Shaft*

	Completed to Jan. 1, 1934, Feet	Completed in 1934, Feet	Total Completed to Jan. 1, 1935, Feet
Drifting	12,663.0	7,480.0	19,143.0
Pilot raising	3,085.0	307.0	3,392.0
Stripping	1,308.0	2,012.0	3,320.0
Steel installation	1,248.0	1,993.5	3,241.5
Ore-pass raises	175.5	1,713.5	1,889.0
Skip-loading stations	0	2	2
Tramway adit	2,053.5	432.5	2,486.0

of the work is shown in a general way in Table 3. Regular hoisting of ore through the Ross shaft began on Nov. 20, 1934, just 11 days before the second anniversary of the day on which it was authorized.

SURFACE PLANT

The general arrangement of the surface plant at the Ross shaft is shown in the plan and elevation in Fig. 14 and the photograph of Fig. 15.

After the site had been selected for the new shaft, it was possible to make adjustments within a very limited area to secure the greatest advantage from surface conditions. The shaft was located at a point on the ridge where excavation to a moderate depth would assure satisfactory footings for foundations and at the same time provide a reasonable working area around the shaft. Advantage was taken of the slope of the hillside to permit a minimum of excavation and provide practically a gravity flow from the skip dump through the crushers to ore-storage bins above the tramway to the mill.

FIG. 15.—SURFACE PLANT AT ROSS SHAFT.

Excavation.—Plans on Nov. 30, 1932, were tentative only and it was necessary to complete and detail all plans before actual work could begin. Grading for a railroad spur to the sites of the headframe and hoist house began on Feb. 23, 1933.

The shaft collar was placed at an elevation of 5355 ft. The top of the ridge was cut down to this level with excavation to a maximum depth of 35 ft. A roughly triangular shaft yard 570 ft. long and 200 ft. wide at the base was thus provided. The hoist-house floor was placed at an elevation of 5290 ft., or 65 ft. below the shaft collar. This necessitated excavation to an elevation of 5280 ft. for basement room and foundations. A yard in front of the hoist house was excavated to basement level. The high bank of the hoist-house excavation is 70 ft. Permanent railroad spurs were laid through the length of the shaft yard and into the end of the crusher building. A temporary railroad spur was laid to and into the hoist building for delivery of material and equipment during construction.

A Bucyrus-Erie 32-B 1-yd. shovel, loading on two 5-yd. Linn tractors, was used for all excavation. The top of the ridge was taken off first, so that sinking of the shaft could be started. As soon as excavation was clear of the shaft, work was begun on the hoist-house excavation, in order that the building and foundations for the hoists might be completed before winter and work of installing the ore hoist might start immediately on delivery. Excavation for the hoist building and machinery foundations was completed on Oct. 4; for crusher and conveyor buildings on Oct. 7; and for the headframe on Nov. 1, 1933. A total of 123,073 cu. yd. of earth and rock was moved, at a cost of 21.8 cents per yard.

Headframe and Crusher Building.—The headframe and crusher building were designed by the Worden-Allen Co. in accordance with plans prepared by Homestake engineers, and material was furnished by that company. The general design is shown in Figs. 14 and 16. The full height of the headframe is 159 ft. Sheaves for the skips are centered at 125 ft. above the collar and those for cage and counterweight are centered at 75 ft. above the collar. A feature of the headframe design is the double back-leg construction with one set of legs for each set of sheaves, which insures a minimum of vibration in operation. The steelwork is covered with 24-gage, galvanized, corrugated copper-bearing steel and is well lighted. All windows have steel sash and ¼-in. factory-hammered wire glass.

Sheaves for skips are 14 ft. and those for cage and counterweight 12 ft. in diameter. They are of cast steel made by the Nordberg Manufacturing Co. All sheaves are mounted in Timken roller bearings. There is an I-beam trolley above and across the upper sheaves, which extends outside the headframe far enough so that the sheave wheels clear the base of the structure when raising or lowering. I-beam trolleys are also placed above each of the lower sheaves, and these may be raised or lowered inside the structure. All working parts of the head-frame are made accessible by steel stairways and platforms.

Skips are of 7-ton capacity and dump into a bin of 275-ton capacity. The top of the bin is 49 ft. 6 in. above the collar. The skip dump is equipped with curved standard 40-lb. rails, with the skip rollers running on the ball of the rail. Bin lining is of cast iron, in sections that may be handled easily.

The crusher house and conveyor gallery are also of steel construction, covered with 24-gage galvanized corrugated copper-bearing steel. The roof is of precast featherweight concrete slab, made by the Federal American Cement Tile Co., covered by five-ply Barrett specification tar and gravel roof. The crusher building is 45 ft. wide by 114 ft. long and 50 ft. high. It is equipped with a four-motor 25-ton Whiting "Tiger" crane with a 5-ton auxiliary hoist traveling the full length of the building.

The railroad spur enters across the north end of the building. Materia for either crushing plant or mine may be unloaded from railway cars by crane.

FIG. 16.—HEADFRAME AND CRUSHER HOUSE AT ROSS SHAFT, DURING STEEL ERECTION, SHOWING GENERAL DESIGN.

Steel erection, as well as all other construction work, was done by Homestake crews. All steel was unloaded as it arrived and stored near the site in such a manner that any piece required was readily available. It was all erected by gin pole. In the upper part of the headframe two to three changes were required before steel was finally placed. The erection time and costs follow:

	Erection Begun	Erection Completed	Total Tons	Erection Cost per Ton
Headframe	10–19–33	3–31–34	274	$25.78
Crusher house	11–19–33	2–28–34	138	17.76

Crushing Equipment.—The primary crushing plant has two identical units, each with capacity equal to the full production from the shaft. In each unit ore from the headframe bin under control of air-operated chute gates is discharged into an Allis Chalmers 8-K gyratory crusher driven by a Westinghouse, type CS totally enclosed 100-hp., 690-r.p.m., 2200-volt, three-phase, 60-cycle motor. A 20-in. belt connects the motor to the driven pulley. The motors are mounted on Rockwood bases. When set at a 4-in. opening, the capacity of this crusher is 150 tons per hour, and with a 6-in. opening it is 250 tons per hour.

The ore discharged from the 8-K crusher passes by gravity, through a grizzly chute, to a 7-ft. Symons cone crusher. The grizzly is made from old stamp stems with maximum openings of 2 in. Each Symons crusher is direct connected to an Allis Chalmers 300-hp., 435-r.p.m., 2200-volt-three-phase, 60-cycle motor. With a feed opening of 14 in. and a discharge opening of 1 in., each crusher has a capacity of 450 tons per hour. Arrangement of crushers is shown in Fig. 17.

The ore that passes through the grizzly and the discharge from the Symons cone crusher drop on to a 42-in. belt conveyor 60 ft. long, which discharges into ore-storage bins.

Pieces of broken drill steel and tramp iron are separated from the ore by means of a Stearns circular suspended magnet, using 250 volt direct current and 3600 watt, and by two Stearns 250-volt direct-current 3993-watt magnetic head pulleys. The suspended magnet is attached to a traveler directly over the discharge chutes of the 8-K crushers and can be transferred from one chute to the other. The magnetic head pulleys are at the bin end of each conveyor.

Ore-storage Bin.—The mill tramway level is 161.4 ft. below the collar of the shaft. Between the conveyor gallery and the tramway level is an ore-storage bin with a capacity of 5000 tons. The dimensions of the bin are 20 by 50 ft. inside, with an average ore-storage height of 100 ft. A 2-ft. reinforced concrete wall divides and reinforces the bin in the center. Diverting chutes are so arranged that ore from either crushing unit may be passed to either section of the bin.

The bin is wholly below surface. It was excavated upward from the tramway level. Regular square sets were used, leaving space for concrete bin walls between the square sets and the surrounding rock. When excavation was completed, concreting of the bin was started at the bottom and carried to the top, working on the square sets and bracing the forms against them. The minimum thickness of the outside bin walls is 18 in. Reinforcing was used in loose ground. The concrete was mixed on the surface and distributed from a 6-in. wrought-iron pipe

placed vertically in the bin, discharging into a chute at the lowest point. As the work progressed, the pipe was shortened by removing lengths as required. In all, 1426 cu. yd. of concrete was required, and its cost was $11 per cubic yard.

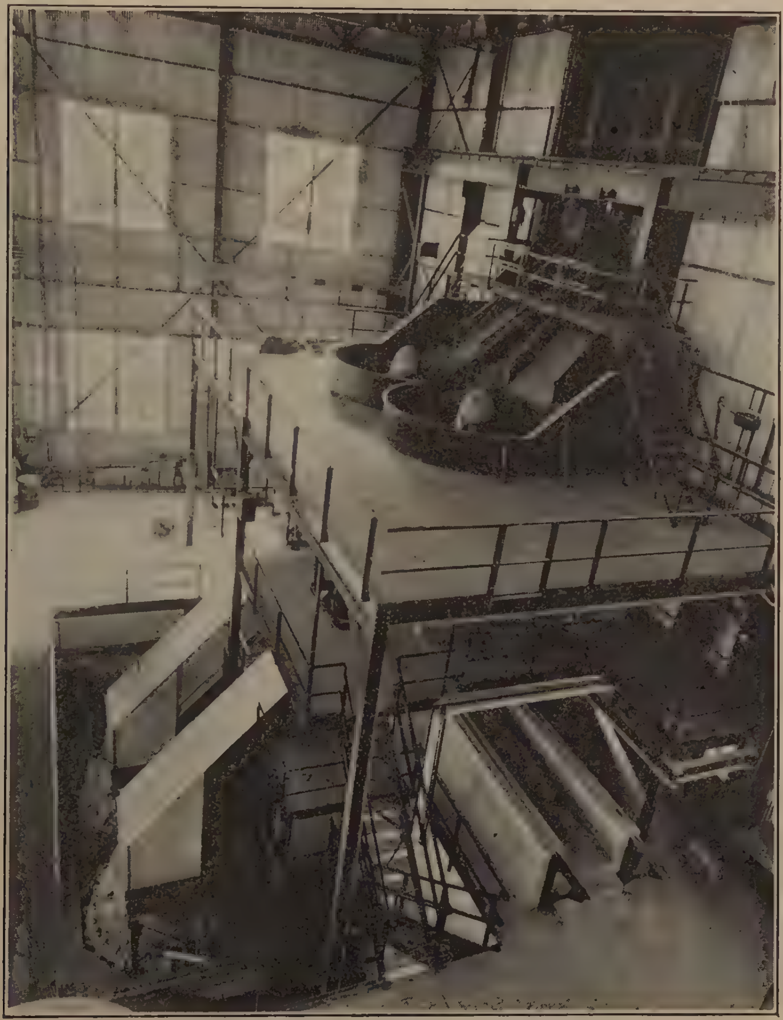


Fig. 17.—Interior view of crusher house at Ross shaft.

The bin walls are lined throughout with slightly worn standard 56-lb. rails placed horizontally and spaced at 9-in. centers. They are held in place by ½ by 3½ by 8½-in. clips fastened by ¾-in. stud bolts set in the concrete. The spaces between the rails will pack with fines, which will protect the clips and bolts and practically limit wear to the ball of the rail. The bottom of the bin is sloped at 45° toward the chutes. Cast-

iron liners 2½ in. thick are used for 12 ft. back from the chutes. Details of the bin lining are shown in Fig. 18.

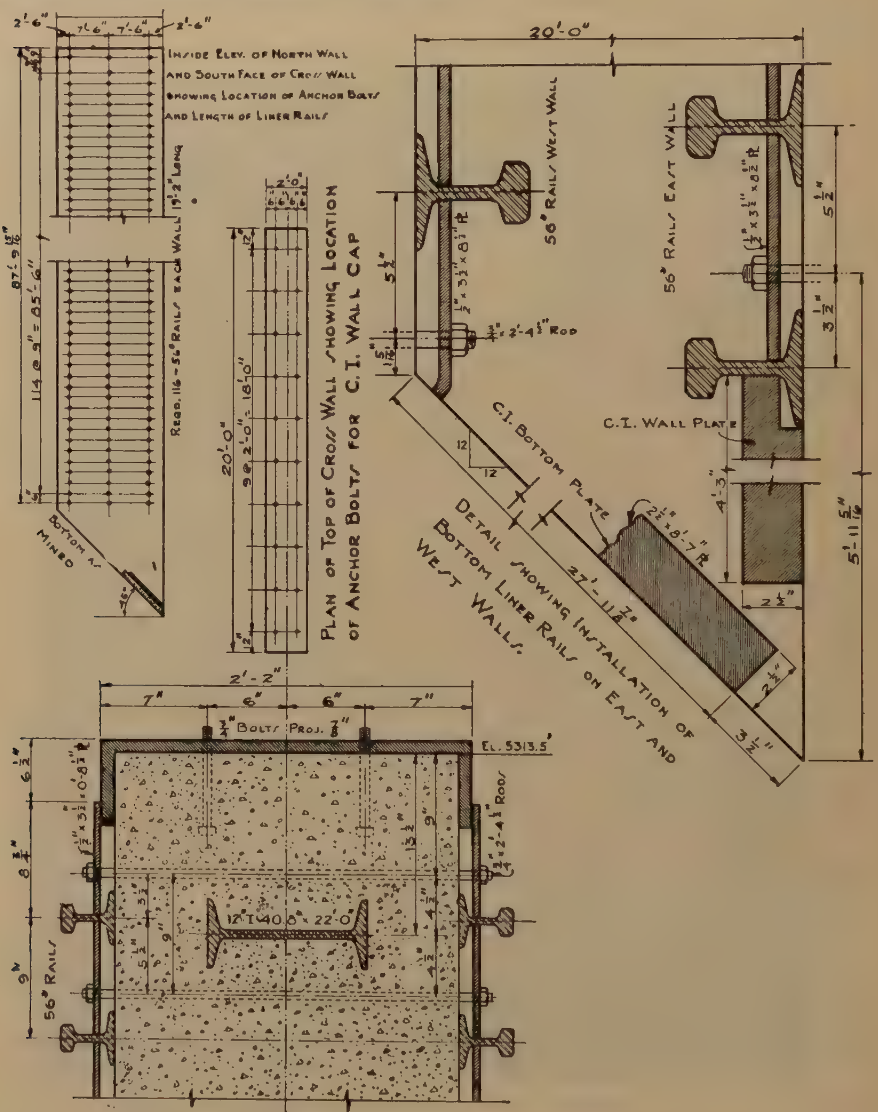


FIG. 18.—DETAILS OF ORE-BIN LINING.

Hoist Building.—The hoist building is located so that the center line of the skip-hoisting engine is on the center line between the skip compartments of the shaft. The center line of the underwind drum of the hoist is 330 ft. and that of the overwind drum 354 ft. from the center line of the shaft. The floor of the hoist building is 65 ft. below the collar of

the shaft. The cage-hoisting engine is set at an angle of approximately 10° with the skip hoist, and in the same building. The motor-generator sets for each hoist are set in opposite ends of the building. A compressor foundation for future use is also in the building. For plan see Figs. 14 and 19.

The hoist building is 80 by 210 ft. and 65 ft. high. It is of brick on a steel frame, roofed with Federal American pre-cast featherweight concrete slabs covered by five-ply Barrett specification tar and gravel

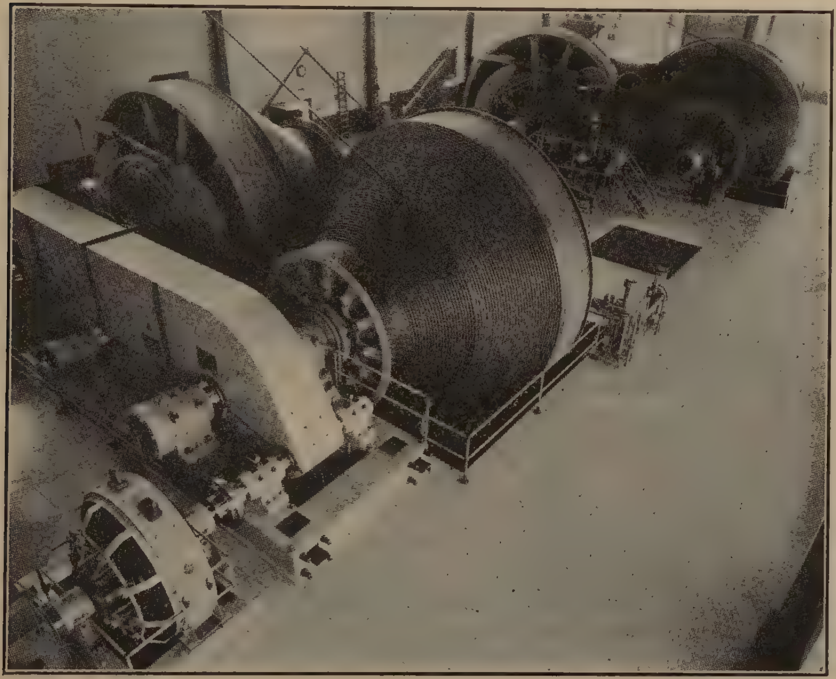


FIG. 19.—INTERIOR OF HOIST BUILDING, SHOWING RELATIVE LOCATION OF HOISTS.

roofing (Fig. 15). The steel work was designed and fabricated by Worden-Allen Co. and erected by Homestake crews. A 25-ton two-motor Whiting "Tiger" crane runs the full length of the building. A temporary railroad spur for use in construction entered the south end of the building.

Building foundations were started Aug. 8, 1933, the building was closed in by Nov. 1 of that year and work begun on the hoist foundations. The building required 688,000 bricks and 265 tons of steel. The cost of handling, erecting and riveting the steel was $12.83 per ton.

Heating Plant and Steam Tunnel.—The Ross hoist building and change room are heated from a central heating plant that is below and adjacent to the hoist building. The building is 32 by 58 ft., of the same

construction as the hoist building. It has a steel and concrete coal-storage bin of 100 tons capacity.

Equipment consists of two Kewanee low-pressure, fire-tube, portable locomotive-type boilers. Each boiler has a capacity of 33,980 sq. ft. of steam radiation. Wyodak subbituminous coal is burned. The boilers

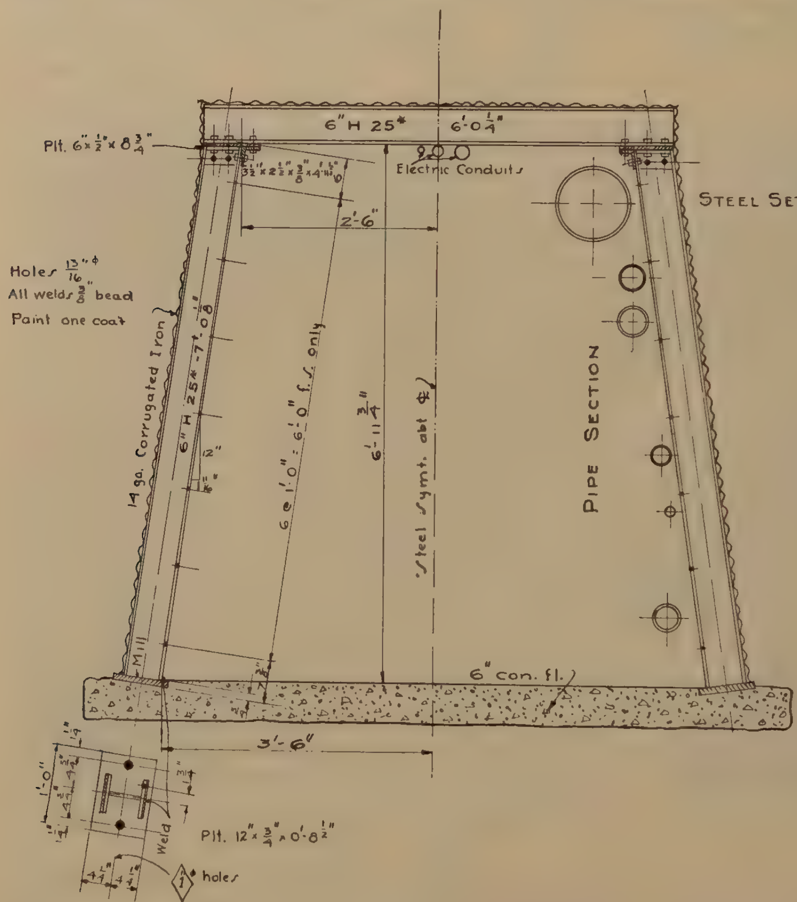


Fig. 20.—Details of steam tunnel.

are fired by two Detroit Underfeed Unistokers, which are equipped with Mercoid controls and forced-draft fans. The breeching is 35 ft. 4 in. long, 36 in. square at the boiler end, and 36 in. by 8 ft. 9 in. at the stack end. The stack is of brick construction, 56 in. inside diameter and 130 ft. high.

Steam and hot-water lines are carried through a tunnel to the hoist building, shaft and change house. The tunnel has steel sets spaced at 5-ft. centers lagged with 14-gage galvanized corrugated copper-bearing steel, and a 6-in. concrete floor. Tunnel details are shown in Fig. 20.

The main header from the boilers is a 10-in. pipe. One 8-in. line branches off this line to the hoist building and another 8-in. line runs to the shaft, up the pipe compartment of the shaft, through the main tunnel to the change house.

The hoist house requires 16,435 sq. ft. of steam radiation, which is furnished through the medium of four Clarage vertical, suspended, unit heaters, each having a capacity of 921,000 B.t.u. per hour, one Sturtevant unit heater having a capacity of 43,100 B.t.u. per hour, and 905 sq. ft. of cast-iron radiation. The height of this building, coupled with the

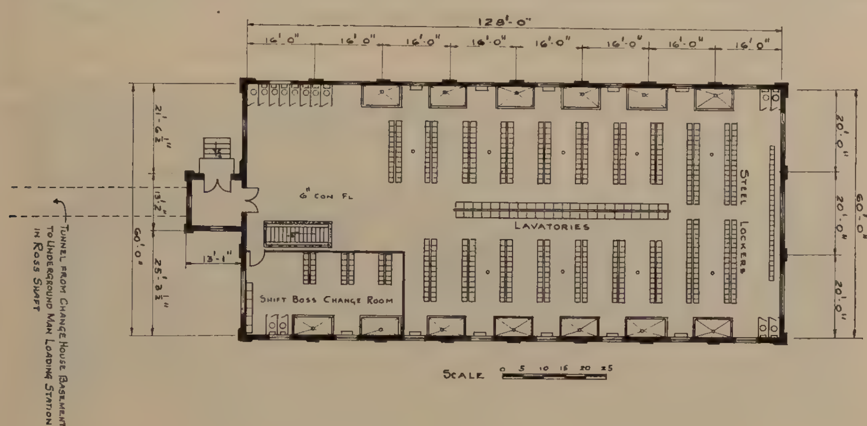


FIG. 21.—FLOOR PLAN OF CHANGE HOUSE.

movement of air caused by the rotating of the large drums of the hoists, may necessitate the use of overhead fans to return to the floor the warm air displaced by the drums.

The change house requires 4400 sq. ft. of radiation, which is supplied by two Sturtevant unit heaters with capacities of 1500 sq. ft. each, two with capacities of 300 sq. ft. each, and 800 sq. ft. of pipe radiation. The change house also requires approximately 7500 gal. of hot water every 24 hr. This is heated by four No. 100 Taco Super-heaters attached to the boilers. The temperature in this building is kept abnormally high in order to dry the damp working clothes of underground workers.

Change House.—The change house is just north of the shaft house (Fig. 14). It is of steel-frame and brick construction, 60 by 128 by 23 ft. high, roofed with Federal American pre-cast concrete slabs under six-ply Barrett specification tar and gravel roof. The building is similar to the hoist house and heating plant.

The change room has 556 lockers on one floor, set in double rows back to back across the building. A double row of lavatories extends down the center and showers along each side. There are 13 shower stalls, each with four shower heads. Ample toilet facilities are provided. A change room for shift bosses is partitioned off in one corner. See Figs. 21 and 22.

There is a basement under one-third of the change room, nearest the shaft. This gives the men who change a few minutes early a place in which to gather before going to the shaft. The basement is connected with the shaft by an inclined concrete tunnel 8 ft. wide by 7 ft. high, to permit travel between the shaft and change room without exposure to the weather. The man-loading station at the shaft is 14 ft. below the collar. The tunnel opens into a room 13 ft. by 26 ft. 9 in. in front of the shaft. In a climate where the thermometer occasionally registers 30° below zero during the winter months, it is believed that the protection thus afforded the men going into and coming out of the mine prevents many cases of cold, grip, etc.

FIG. 22.—INTERIOR VIEW OF CHANGE HOUSE.

The carbide and oil room is just off this tunnel, next to the basement of the change room. There is a counter height opening between this room and the tunnel for issue of supplies to the men. A lamp room, 14 by 25 ft., of brick construction is in the basement of the change house. It is equipped for storing and recharging electric lamps. Windows are provided for receiving and issuing lamps to miners.

All buildings in the Ross shaft group are of 100 per cent fireproof construction. The only timber in the structures is in the guides in the headframe.

THE HOISTS

The hoist installation at the Ross shaft is of particular interest because the hoists are a radical departure from the conventional design of hoists in use for large-scale hoisting from great depths in the Western Hemisphere, and also because they are among the largest hoists ever built. In physical size they are the largest motor-driven hoists ever built in this part of the world. The design of the hoists was developed by the engineers of the Nordberg Manufacturing Co., to meet the following definite conditions specified by the Homestake engineering and operating staff:

1. That the drums should be large enough to wind in one layer the ropes for the ultimate depth contemplated, in order to reduce rope wear.
2. That the hoists should be placed as near the shaft as possible.
3. That there should be all possible duplication of identical parts. This was attained particularly in the electrical end of the equipment.
4. That power peaks must be kept at a minimum.

As a general rule, very large hoists have only one drum, either cylindrical, conical or cylindroconical. If they are counterbalanced, one rope is wound on to the drum while the other is wound off. Such hoists are not well adapted for multilevel hoisting. As ore is being hoisted at the Homestake from all levels, from the 300-ft. level to the deepest opened, it is common practice to use hoisting machines with double drums, and the new hoists are of that kind.

The proposal that the individual drums be large enough to wind 5400 ft. of 1⅞-in. rope in single layers was met with amazement, because it was so radically different from general practice with electric motor-driven hoists in this country. This plan was also questioned because these large machines were to be driven by a comparatively small power system. It was fully realized that such large hoists would be costly, but it was estimated that the saving in wear on ropes would be sufficient to repay the increased cost. It soon became evident to all concerned that the type of machine tentatively selected would be well suited for the hoisting duty required. It also developed that, while the large winding ends of these machines cost considerably more than those of conventional design, the total cost was not so much greater, because the shape of the drums produces a duty cycle that reduces the size of the electrical apparatus required.

The specifications submitted to the manufacturers were as follows:

Ore Hoist

Double-compartment shaft, vertical lift.
Electric drive, geared.
Double drum, both clutched.
Axial plate clutches, double disk, multiple arm.
Post brakes.
Rope diameter, 1⅞ inches.
Single wrap or layer of rope preferred.
Length of active rope, 5400 feet.
Rope speed, 2500 feet per minute.
Hoisting duty: ore, 14,000 lb.; skip, 12,500 lb.; rope, 5400 ft. at 5.63 lb., 30,400 lb. total, 56,900 lb.
Ultimate depth of shaft, 5220 feet.
Distance collar of shaft to head sheaves, 125 feet.
Time cycle: acceleration, 15 sec.; loading, 6 sec.; deceleration, 15 sec.
Hoist to be capable of lifting an occasional emergency trip unbalanced.
Electric current available, 2200 volts, three-phase, 60-cycle.
Direct-current hoist motor with flywheel motor-generator set and Ward-Leonard control.

Complete safety appliances required.

Minimum tonnage: output 3000-ft. hoisting depth in 15 hr., 4000 tons; output 4000-ft. hoisting depth in 15 hr., 3000 tons.

Man and Material Hoist

Double-compartment shaft; one for cage and one for counterweight. Vertical lift.

Electric drive, geared.

Double drum, both clutched.

Axial plate clutches, double disk, multiple arm.

Post brakes.

Rope diameter, 1⅝ inches.

Single wrap or layer of rope preferred.

Length of active rope, 5400 feet.

Rope speed, 2500 ft. per minute.

Hoisting duty: cage, 8000 lb.; load, 10,000 lb.; rope, 5400 at 4.23 lb., 22,840 lb.; total 40,840 lb.

Ultimate depth of shaft, 5220 feet.

Distance collar of shaft to head sheaves, 75 feet.

Time cycle: acceleration, 20 sec. loading two cars, 30 sec. deceleration, 15 sec.

Hoist to be capable of lifting an occasional emergency trip unbalanced.

Electric current available, 2200 volts, three-phase, 60-cycle.

Direct-current hoist motor with flywheel motor-generator set and Ward-Leonard control.

The equipment shall be proportioned to handle safely the following duty cycles:

1. Hoisting rock, with counterbalance in action, from any of the lower levels up to the 2500-ft. level. This rock hoisting to range in lift from 500 to 2500 feet.
2. To make an occasional trip in counterbalance either up or down the entire depth of the shaft with a 10,000-lb. load.
3. To operate continuously at run-around work with an average load of 2000 pounds.
4. To lower and hoist men at beginning and end of shifts between all levels and the surface. The load will consist of 35 men, or a total of 5600 pounds.
5. To hoist men in an emergency without counterweight at reduced speed with full loads.

Mechanical End of Hoists

The two hoists, as designed and built by the Nordberg Manufacturing Co., are identical except that the drums are grooved for their respective rope sizes and the ore hoist is driven by two motors and the man and material hoist by only one.

Each hoist has two bi-cylindroconical drums, each mounted on a separate shaft. The shafts are parallel and the drums are in tandem. This design was developed to reduce the fleet angle, so as to permit placing these large hoists reasonably near the shaft. Each drum has three distinct parts: a small cylindrical section 12 ft. in diameter, a large cylindrical section 25 ft. in diameter, and a frustum of a cone connecting the two cylindrical parts. The width between the flanges is approximately 17 ft. See Fig. 19.

The small end of the drum is of cast steel, in halves, each of which weighs 19,880 lb. This part of the drum holds 15 turns, or 565 ft. of rope. The conical part of the drum is of cast iron, in 12 identical sections each of which weighs 5050 lb. It holds 52 turns, or 2835 ft. of rope. The large cylindrical end of the drum is of cast iron in six sections, each weighing 6120 lb. It holds 27 turns, or 2000 ft. of rope. All sections of the drum are flanged with machined joints and bolted with turned bolts fitted into reamed holes. The grooves on the drums are machined thread grooves. The ore hoist has 2-in. grooves for 1⅞-in. rope. The man hoist has 1⅞-in. grooves, allowing for possible future use of 1¾-in. rope. The lengths of rope on each part of the drums, mentioned above, refer to the drums on the ore hoist. On the man and material hoist the cylindrical parts hold slightly more rope.

The drum shafts are 40 ft. 5 in. long, 30 in. in diameter and weigh 77,000 lb. each. Each shaft has a 6-in. hole drilled through its entire length—drilled to insure solid metal and utilized for the introduction of oil to the drum shaft bearings. Each shaft is enlarged where the drum spider bushings run on the shaft, and also for the fit of the main driving gears.

The four main bearings are 24 in. by 48 in. long. The two outboard bearings are 20 by 36 in. long and have marine-type thrust bearings to take care of the lateral motion of the shafts. The idler shaft bearings are 16 in. by 26 in. long and also have marine-type thrust bearings. The pinion shaft bearings are 12 in. by 24 in. long. All bearings are of the two-part removable-shell type and lubricated by oil under pressure.

The clutches are 14-ft. axial plate, friction type, lined with molded asbestos blocks. Each is operated by a double-acting hydraulic cylinder with 8-in. bore and 14-in. stroke. See Fig. 19.

The brakes are parallel-motion, post type. They are applied by gravity but released by a single-acting hydraulic cylinder with 7-in. bore and 24-in. stroke. The brake blocks are of basswood.

The hydraulic system consists of a sump, pressure tank, two Northern gear pumps and two motors. The sump and pressure tank have capacities of approximately 225 gal. The pumps have capacities of 20 gal. per min. each, and are direct connected to General Electric 3-hp., 1155-r.p.m. motors. One pump is sufficient to operate the hydraulic system; the other is a spare.

At one end of the drum shafts a train of gears connects the two shafts—a main gear on each drum shaft meshing with an idler gear between. The two main gears and the idler gear are identical. They are continuous tooth herringbone gears with solid rims and split hubs. They have 215 teeth, 1½ diametrical pitch, 143.333-in. pitch diameter, and 30-in. face. These gears weigh approximately 25,900 lb. each. The pinions are forged integral with the pinion shaft. They have 25

teeth, 1½ diametrical pitch, 16.666-in. pitch diameter and 30-in. face. The pinions are connected to the motors by Falk flexible couplings. The ore hoist, being driven by two motors, has two pinions, one at each end of the train of gears. The man and material hoist has only one motor and one pinion.

All shaft bearings are lubricated by an automatic oiling system, which consists of a bag-type oil filter that acts as a sump and duplicate motor-driven centrifugal pumps. The filter is of Wm. H. Nugent & Co. manufacture and has a capacity of approximately 205 gal. The pumps are Dayton-Dowd centrifugal direct connected to two General Electric ¾-hp., 1735-r.p.m. motors. An automatic grease lubricator for bushings is mounted on the inside of each 12-ft. spider.

Safety devices on the hoists consist of two model C Lilly controllers, a Nordberg slow-down device, limit switches, and a safety interlocking arrangement, the working of which makes it impossible for the operator of the hoist to release the clutch unless the respective brake is fully applied. The Lilly controllers prevent overspeeding and overwinding, and regulate the application of the brakes when they are automatically applied in an emergency. The Lilly controllers also warn the operator by an electric bell when the full normal speed is exceeded. The slow-down device, a Nordberg feature, has a "slow-down" or cam turn-off, which operates on the principle of a traveling nut and is connected with the control lever to effect automatic retardation. The limit switch is an electrical device above the collar of the shaft, which, when a skip or cage is traveling too far in the direction of the headsheaves, breaks a connection and shuts off the power to the hoist motors and sets the brakes.

The drums make 35 revolutions per minute. The maximum rope speeds on the ore hoist are 1320 ft. per min. on the 12-ft. diameter of the drums and 2750 ft. per min. on the 25-ft. diameter. The rope speeds on the man hoist are the same as those on the ore hoist, with a controlled maximum of 800 ft. per min. for hoisting men. Acceleration time on the ore hoist is 15 sec.; deceleration time is also 15 sec. Acceleration time on the man hoist is 20 sec.; deceleration time is 15 seconds.

The ore hoist is equipped with Roebling 1⅞-in. special Seale construction Blue Center steel rope. This rope weighs 5.63 lb. per foot and has a breaking strength of 142 tons. The man hoist is equipped with 1⅝-in. Leschen special Seale construction Hercules "Red Strand" steel rope. This rope weighs 4.23 lb. per foot and has a breaking strength of 108 tons.

The following data and Table 4 may give the reader a clearer idea of the size of these hoists:

Twenty-three railroad cars were required to transport each hoist, exclusive of electrical equipment, from the factory to Lead. Each hoist

weighs 1,040,000 lb. Each hoist occupies a floor area approximately 56 by 61 feet.

TABLE 4.—*Relative Location of Head Sheaves and Drums of Hoists*

	Ore Hoist	Man Hoist
Elevation of sheaves, ft.	5475	5425
Elevation of hoist house floor, ft.	5290	5290
Difference in elevation, ft.	185	135
Distance between center of drums and center of sheaves, ft.:		
West drum to north sheave	396.75	381.63
East drum to south sheave	357.6	357.63
Maximum fleet angles:		
West drum to north sheave	1° 10′	1° 21′
East drum to south sheave	1° 10′	1° 27′

Electrical End of Hoists

The electrical equipment for the Ross hoists was furnished by the General Electric Co., designed to conform with conditions specified by Homestake electrical engineers.

The hoists are gear-driven by direct-current motors for which power is furnished by motor-generator sets equipped with flywheels. This driving and control system as a whole is known as the Ilgner-Ward-Leonard system, and, briefly, consists in manipulating the shunt field of the generator to give motion and direction of rotation to the hoist motor.

As stated, the man hoist is driven by one motor and the ore hoist by two motors. Each motor has a capacity of 1500 hp. They are exact duplicates and therefore are interchangeable as a whole or in parts. There were several reasons for equipping the ore hoist with two motors instead of one. Power calculations for the hoists developed the fact that the ore hoist required 3000 hp. and the man hoist 1500 hp. With two motors on the ore hoist, all three motors could then be of the same size and identical throughout, which minimized the number of spare parts needed. If desirable, a complete armature and field structure could be kept as spares. It also improved the gearing arrangement on the ore hoist, as the work was equally divided between two pinions. One of the best features of the two-motor drive is that one motor is large enough to drive the hoist when hoisting from the bottom level (5200 ft.). That makes it possible to hoist from that level if a generator or motor is damaged.

The motor-generator set for the ore hoist has two generators of 1250-kw. capacity driven by a 1750-hp. motor with a steel-plate flywheel weighing 87,500 lb. (Fig. 23). That for the man hoist has one generator of 1250-kw. capacity driven by an 800-hp. motor with a flywheel weighing

67,000 lb. Both sets run at the same synchronous speed; 720 r.p.m. The three 1250-kw. generators are identical and all parts are interchangeable. The motors are of the wound-rotor induction type.

The flywheels are necessary on the motor-generator sets for these hoists because the Homestake power system is too small to stand the great and violently fluctuating power demands produced by them. For example, while hoisting ore from a shallow depth the power output of the hoist motors is plus 4200 hp. at the end of the accelerating period and minus 1500 hp. at the beginning of retardation, or a total variation of 5700 hp. Even when hoisting from the ultimate bottom level, the power variation will still be a total of 4100 hp. The flywheel of a motor-

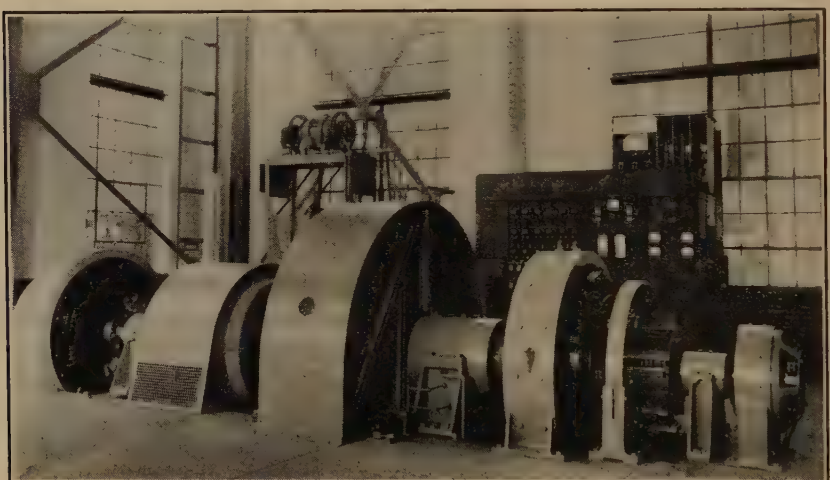


FIG. 23.—MOTOR-GENERATOR SET FOR ROSS ORE HOIST.

generator set is used to equalize the power fluctuations produced by the hoist and thereby cause the power demands on the power system to be reasonably uniform. This result is attained by automatically limiting the maximum power input to the motor of the motor-generator set to the average required to perform a given hoisting cycle. The varying demands of the hoist motor are then taken care of by the exchange of energy with the flywheel through speed variations.

The generators and motors of the ore-hoist equipment are connected in alternate series; i.e., the connection from the first generator is to the first motor, then from that motor to the second generator, then to the second motor, and back to the first generator. This form of connection limits the maximum voltage between any parts of the circuit to the voltage of one generator (600 volts) and between some other parts of the circuit there is zero voltage.

The hoist control is semiautomatic. To put the hoist in motion the operator may move the control lever to the full-on position at once and

the machine will come to full speed with normal acceleration. In stopping the hoist, the control lever may be moved to the off position instantly and the machine will stop gently. But the operator must apply the brakes to hold the hoist in the stopped position. It is also possible to instantly move the control lever from the full-on position in one direction to the full-on position in the opposite direction without causing shock to the apparatus. Where such a move is made the hoist will gently come to rest and then accelerate to full speed in the other direction. Apart from the safety feature this automatic accelerating and decelerating is of great value, for hoisting can be safely done at a much higher rate of speed because it relieves the operator of the duty of having to accelerate or retard.

Idlers for Hoist Cables

Final decision has not yet been reached as to whether or not idlers will be necessary between hoist drums and sheaves. The spans range from 357 to 396 ft. No idlers have yet been installed. It is hoped that none will be necessary, as they only add to rope wear. The ore hoist has been in regular operation for two months, raising ore. Prior to that it was in operation for shaft construction work for seven months. The man hoist has been in operation for shaft construction work for four months. The sag in the cables is large, particularly when the empty cage or skip is at the top. It is greatest as the empty skip is coming out of the dump. The operation without idlers, however, is sufficiently satisfactory to allow time for thorough testing over a long period before final decision must be made. At present it is our opinion that idlers will not be necessary.

Operation of Ross Shaft

The first ore was hoisted at the Ross shaft on Nov. 19, 1934. On that day 36 skips were raised, to test and check all parts of the equipment. On the following day regular hoisting began and has continued since without interruption. Hoisting at capacity has not yet been continued for a full shift, but 150 skips, or 1050 tons, have been raised in 4 hr., which equals the estimated capacity for that period. From results to date it is quite evident that the hoists will do more than is expected of them. Their performance has been very gratifying.

Ore has been hoisted from two loading stations only, that above the 2150-ft. level and that above the 2750 level. These are 2300 ft. and 2900 ft., respectively, below the skip dump. This hoisting gives the most severe duty for which the machine was designed but it is handling the work without the slightest sign of distress. At present the hoist is favored to some extent in hoisting from shallow depth, because the hoisting ropes are only 4000 ft. long. This gives greater advantage

of the conical part of the drums than can be obtained with full-length cables in hoisting from the same depths. Present ropes will have served their useful life before longer ropes will be required. On the basis of present performance no trouble is expected when hoisting from shallow depth with full length ropes.

On the basis of operation over a brief period, it is our conclusion that this type of hoist is far superior to those with cylindrical drums winding large ropes in many layers. We believe that this design will gain favor for heavy-duty hoisting from great depth.

Concreting Drifts at Ray Mines Division of Kennecott Copper Corporation

By Robert W. Thomas,* Member A.I.M.E.

(New York Meeting, February, 1937)

During the past 20 years the advantages of reinforced concrete as a substitute for timbering in so-called permanent mine openings have been fully recognized, and its use has become almost general practice in the larger mines. The term "permanent mine openings" is used advisedly and is assumed to embrace such openings as ventilation shafts, material shafts and main hoisting shafts, with their adjacent stations, ore pockets, etc. In the concreting of such openings, the actual cost as compared with that of timbering has been of secondary consideration, the support of large openings, elimination of fire hazards, and prevention of interruptions to production being the primary objectives. The ultimate economy, however, of lining such openings with concrete when they are to remain in use during the entire life of the mine, and especially when they are in bad ground, will seldom be questioned.

During this 20-year period there have appeared in engineering publications a number of interesting articles on the use of concrete in mines, nearly always describing some specific project of major proportions, such as the concreting of ore pockets, large stations or main hoisting shafts. Such articles are unquestionably of interest, since the projects require considerable engineering thought, not only in the method of excavating large openings, but in their engineering design as well. Concreting of main hoisting shafts, particularly when this is done with the least possible interruption to production, provides some interesting problems. The concreting of haulage drifts of the dimensions commonly found in mines, on the other hand, soon develops into a routine operation, therefore it is with some hesitancy that the author attempts to prepare an article on this subject. The writer can recall only one article on the concreting of mine haulageways, although several articles have been written concerning the concreting of tunnels considerably larger in dimensions than those required in general mine practice.

From the scarcity of articles on the subject, it would seem that the concreting of haulage drifts is still far from being considered general

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practice, which is difficult to explain, because some main haulage drifts undoubtedly fall into the category of permanent mine openings, and should be given the same consideration, both as to the elimination of the fire hazard and of interruption to mine traffic. Furthermore, the introduction of the pneumatic concrete placer, which has greatly simplified the work of lining drifts, regardless of their length or small dimensions of the working space, has made it possible for this work to be done with speed and economy. For these reasons it seems logical to give concrete serious consideration whenever geological conditions or previous experience indicates the necessity of a complete retimbering job during the useful life of the drift.

Concreting at Ray Mines

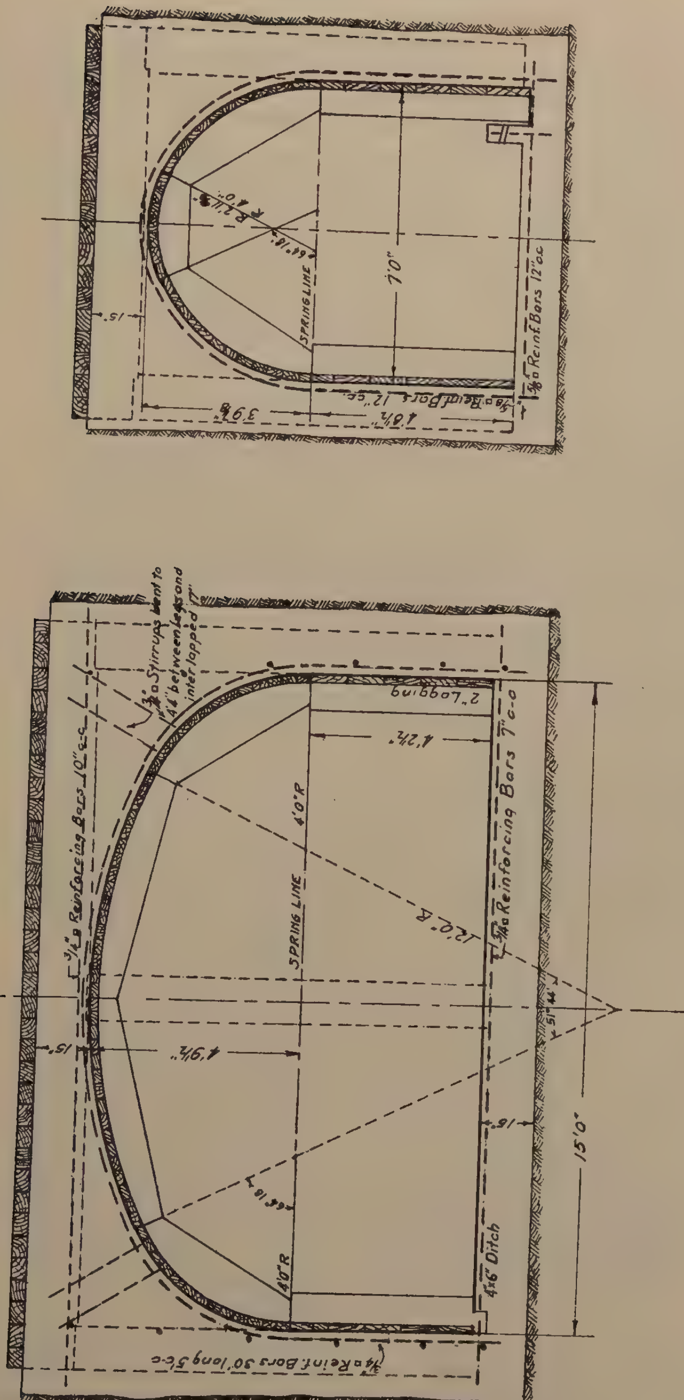
At the Ray mines experience has been gained in the concreting of some 6000 ft. of single and double-track drifts, and this paper will be confined to a résumé of some of the larger projects that have been undertaken and completed during the past 20 years. In each of the projects to be described the concrete was mixed on the surface and delivered by gravity through a 6-in. pipe to the pneumatic placer below, from which it was forced through a 6-in. pipe directly into the forms in the tunnel.

Little need be said regarding the mixing plant on the surface, other than that all gravel bins were placed so that gravel could be delivered directly from chutes or trammed in cars to the mixer, which was placed at the collar of the shaft.

Third Level

The first work of this nature was started late in 1916 on the third level of the No. 2 shaft. In this particular instance a haulage drift some 1600 ft. long from the main hoisting shaft to the mining area was giving considerable trouble because of the swelling nature of the ground. Constant timber repairs, together with the necessary removal of the slacked ground, greatly aggravated this situation, and it was finally realized that to maintain the drift without constant recurrence of delays to production a more satisfactory lining than the usual timber sets would have to be installed. Experiments were made involving the use of a thin seal of concrete between the lagging and the rock walls, which, it was thought, would exclude the air and stop its slacking action on the ground, but eventually it was decided that a complete lining of reinforced concrete would be necessary to meet all the requirements. This would not only exclude the air but would also support the ground.

Twelve by twelve-inch timber sets on 5-ft. 0-in. centers were already in place, and because of the extremely heavy nature of the ground it was deemed advisable to leave these timbers within the concrete. As it was



SINGLE TRACK SECTION

DOUBLE TRACK SECTION

FIG. 1.—CONVENTIONAL ARCH DESIGN FOR CONCRETING HAULAGEWAYS.

necessary to maintain the same clearance as existed in the original sets, the posts had either to be set back or cut back, so that they would not interfere with the construction of the forms. The bottom of the tunnel was first prepared by replacing the timber sills with small spreaders and then excavating the floor to allow for the desired thickness of reinforced concrete. A section of bottom was then poured, leaving reinforcing rods extending upward as stubs for the side walls. The side walls and back were next poured successively in 10-ft. sections. The conventional arched back design, as shown in Figs. 1, 2, 3 and 4, was used, the forms being constructed of 2-in. material supported by 3-in. studding and arch segments spaced on 2-ft. 6-in. centers.

Fig. 2.—Concreted haulageway, arch design.

At first the bottom and side walls up to the spring line of the arch were placed by hand, the concrete being trammed from the receiving station at the shaft, and only the arched back section was placed pneumatically. The reason for this was that the first attempts to use the pneumatic placer in filling the entire form resulted in rather marked segregation of the mixture, and it was thought that the segregation was due to the long distance from the placer to the form—considerably over 1500 feet.

On this project a total length of 1260 ft. of single-track tunnel and 565 ft. of double-track tunnel was completed at an average cost of $25.24 per cubic yard. This amount does not include the driving or timbering of the original drift, but does include the considerable cost of realignment, together with the necessary excavation and timber repairs.

Fourth Level

In 1924 further development at the No. 2 shaft made necessary the opening of the fourth haulage level, and, as the main haulage artery of

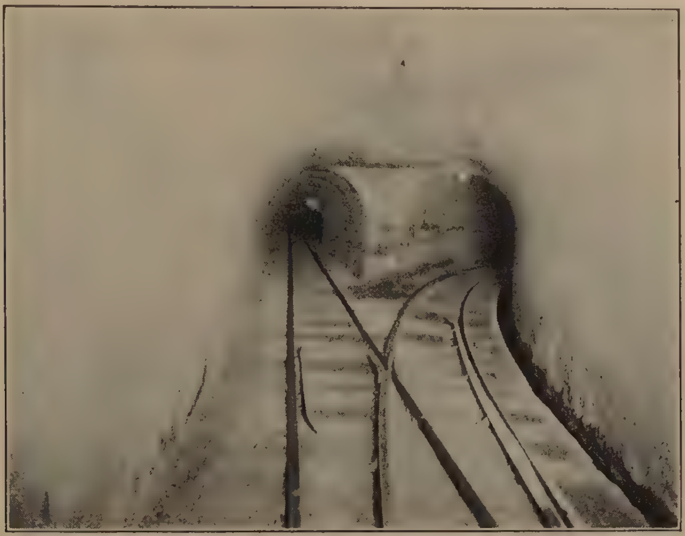


FIG. 3.—CONCRETED HAULAGEWAY, ARCH DESIGN.

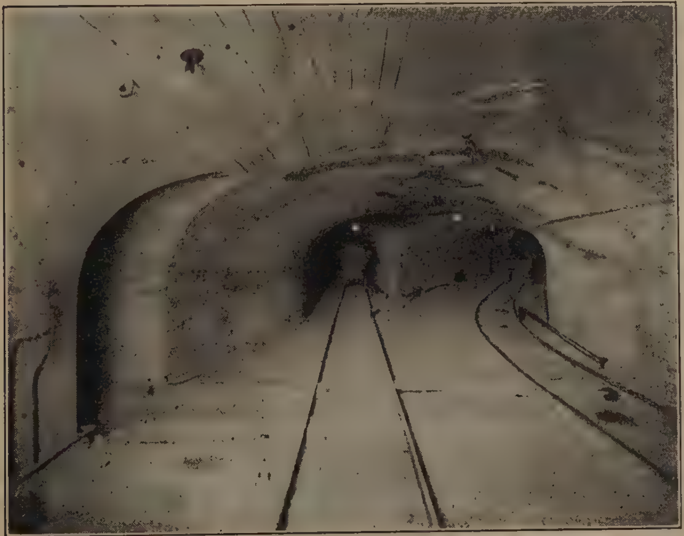
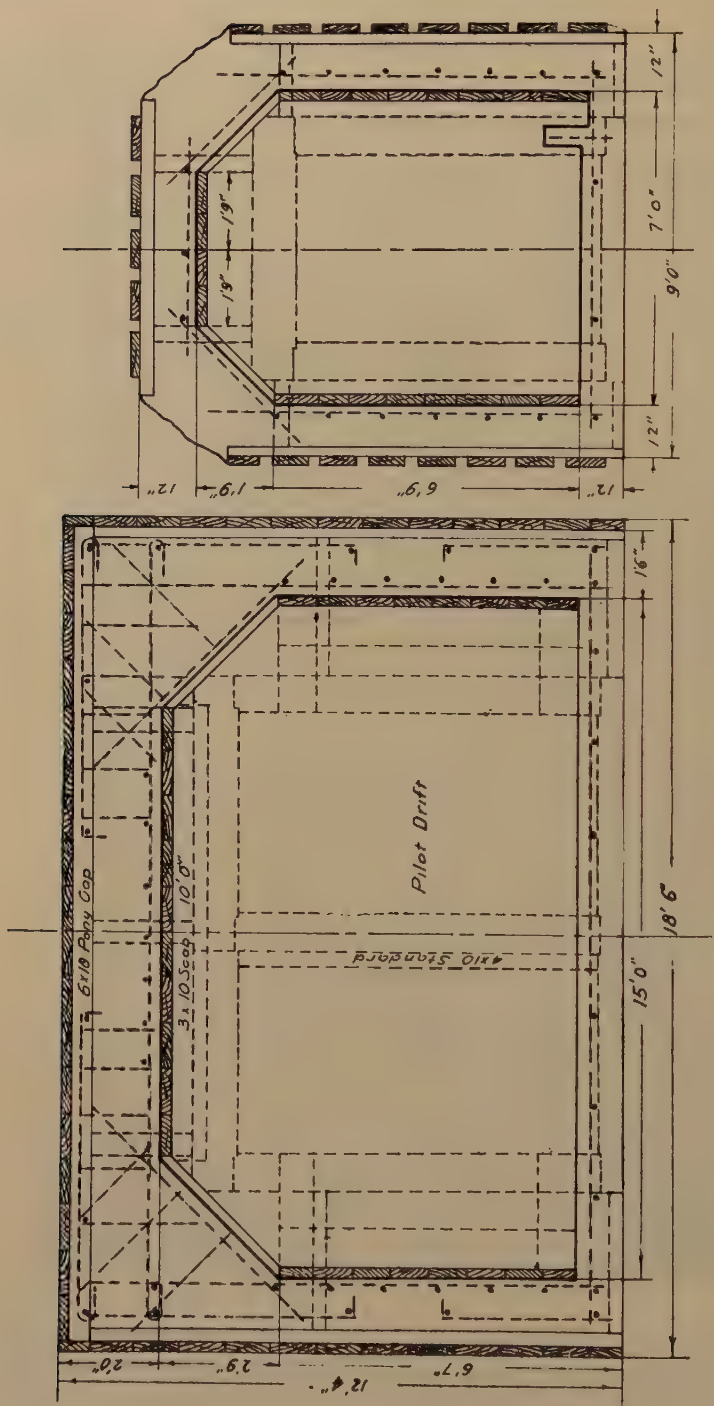


FIG. 4.—CONCRETED HAULAGEWAY, ARCH DESIGN.

this level would pass through the same type of ground as had been encountered on the third level, it was evident that some considerable footage of this drift would have to be lined with concrete.



SINGLE TRACK SECTION

DOUBLE TRACK SECTION

FIG. 5.—BATTERED (KNEE-ARCH) DESIGN FOR CONCRETING HAULAGEWAYS.

Taking advantage of the experience gained in concreting the third level, several changes were decided upon, which materially simplified the work and reduced the costs. On the third level, since the drift had already been driven and timbered to required width, it was advisable to use the existing timber as ground and form support during the work of concreting, a procedure that necessitated leaving the original timber within the concrete. It was decided that in the project now to be undertaken a minimum of timber should be left within the concrete, and this was accomplished in the single-track tunnel by first driving a pilot drift, the timber sets of which were of such size that their outside

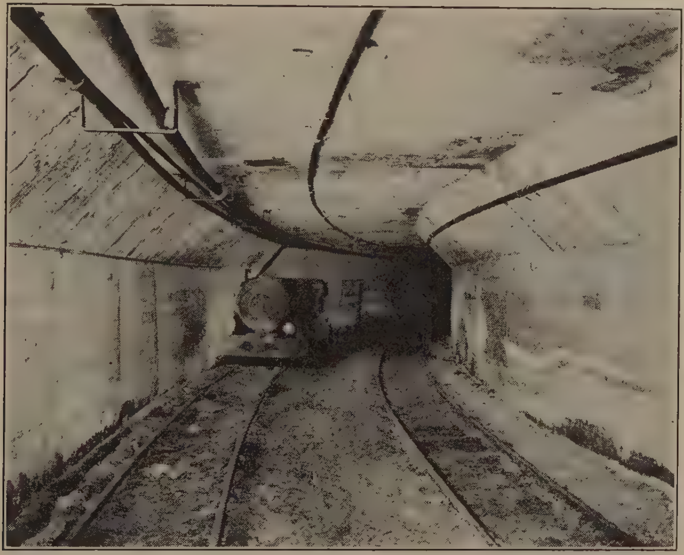


FIG. 6.—CONCRETED HAULAGEWAY, BATTERED (KNEE ARCH) DESIGN.

dimensions gave the proper supports for the concrete forms without encroaching on the space to be filled with concrete. It was also decided to simplify the sectional design, and instead of the conventional arch type of tunnel, which required expensive framing of form material and more skillful labor, the so-called battered (knee-arch) section shown in Figs. 5, 6, 7 and 8 was used.

As soon as the pilot drift was completed, the work of enlarging or trimming the rock sides, roof and floor to required size was started, progressing just in advance of the placing of the concrete. The floor was poured first, with stub reinforcing for bond to the side walls. The wooden forms were then erected and the concrete side walls and back poured in one operation, each pour representing an advance of 10 ft. along the drift. The reason for this regular and close time sequence in the several steps of the work was to get the concrete in place as soon as possible after the rock surfaces had been prepared to receive it, thus minimiz-

ing the slacking effect of the air on the freshly exposed surface of the rock. In the double-track section the same procedure was followed, with some modification in the work of enlarging, as shown in Fig. 5.

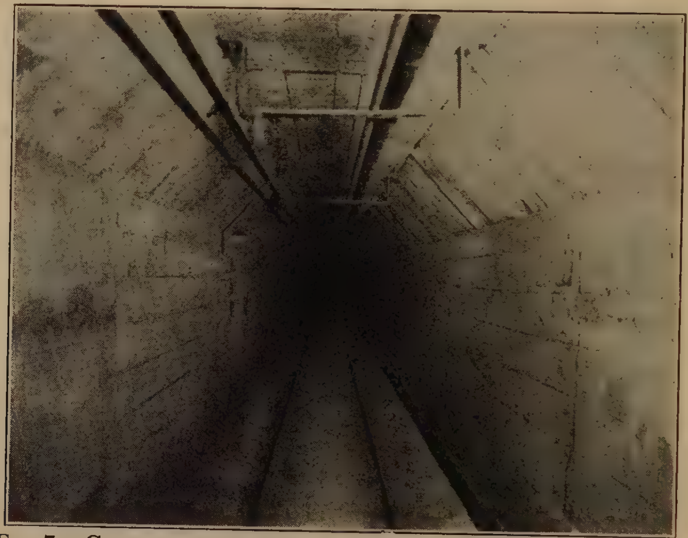


FIG. 7.—CONCRETED HAULAGEWAY, BATTERED (KNEE ARCH) DESIGN.

FIG. 8.—CONCRETED HAULAGEWAY, BATTERED (KNEE ARCH) DESIGN.

Because of the simplified form work and the improvement in general performance gained from previous experience, the crew on this job was able to average 10 linear feet of 7-ft. 0-in. by 8-ft. 6-in. tunnel per day of two 8-hr. shifts, as compared with 10 linear feet per day of three 8-hr.

shifts on the third level project. The concrete work on this level consisted of 780 ft. of single-track and 520 ft. of double-track tunnel. The cost of the concrete poured, including all excavation, form building, etc., but exclusive of the original cost of driving the pilot drifts, averaged $22 per cubic yard.

Drain Tunnel

The third and most extensive concrete lining work was completed in 1925 in the Sonora drain tunnel. This was a drainage drift 3344 ft. long, driven in 1913 and 1914 to protect extensive caving operations from flood waters, which had required considerable maintenance work in the succeeding years. This tunnel, which was driven approximately 8 ft. 6 in. by 9 ft. 0 in. in the clear and in comparatively good ground, was timbered throughout the major portion, not on account of heavy ground but rather to prevent occasional sloughing, and at the same time to provide support for timber lining. The repairs necessary in maintaining this tunnel resulted from damage to the lining by flood waters and debris which the tunnel carried during violent storms. Upon decision

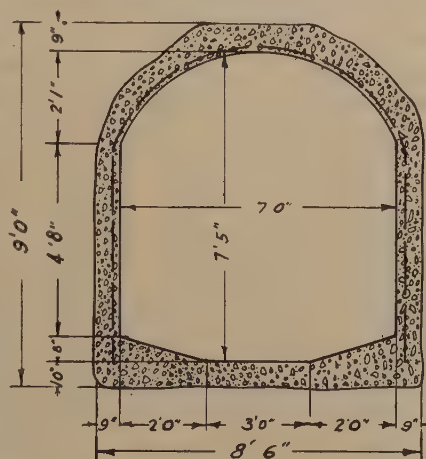


FIG. 9.—TYPICAL SECTION OF DRAINAGE TUNNEL.

to concrete this tunnel, a catchment basin approximately 600 ft. from one end of the tunnel was chosen as the most satisfactory location for the mixer and pneumatic placer. The mixer on the surface delivered the concrete to the placer, which was about 15 ft. below.

In order to increase efficiency in form work sectional steel forms were fabricated. These were made of light-weight plate on angle-iron frames and were fitted with lugs for locking in place. The steel sections were 5 ft. long, and all units were light enough for two men to put in position. The finished concrete in most of the tunnel was 7 ft. 0 in. wide by 7 ft. 5 in. high, with a drain bottom as shown in Fig. 9. On this job as much as 250 ft. of bottom was prepared and poured in advance, leaving reinforcing stubs extending for the side walls. After a stretch of bottom was thus prepared, the crew had no difficulty in concreting an average of 30 ft. of tunnel per day of two 8-hr. shifts. This exceptional advance, as compared with previous jobs, was due, as mentioned above, mainly to the condition of the tunnel. Since no enlarging was required, it was necessary only to remove the timbering, pour the concrete bottom, erect

the steel forms, and pour the remainder of the concrete. In the lining of this 3344 ft. of tunnel the cost of concreting amounted to $22.20 per cubic yard.

Practical Problems in Concreting Mine Drifts

The lining of mine drifts with concrete presents no particular engineering problem, but rather gives rise to numerous practical problems, which must be solved as the work progresses. A few of these problems are discussed in the following paragraphs.

Segregation of Mix.—As has been mentioned, the first attempt to place concrete with the pneumatic placer where the distance from the placer to the form was over 1500 ft. resulted in rather marked segregation of the mix, and the difficulty was believed to be due entirely to excessive length of the 6-in. delivery pipe. As a matter of fact, this conclusion was erroneous, as in subsequent work the segregation was greatly reduced when shooting distances as great as 2500 ft., and it was demonstrated that there were several causes contributing to segregation, all of which could be overcome to a great extent. One important factor was the point at which the discharge pipe entered the form. In the first attempts to pour the side walls and back of the tunnel in one operation, the pipe discharged directly over the center of the arch, and the concrete ran or flowed over the top of the form toward each side, with resulting alternate layers of coarse and fine components of the aggregate as it came to rest in the side walls. It was found that this trouble could be considerably lessened by placing the discharge pipe over each of the side walls successively and finally over the back.

Another factor influencing segregation was the consistency of the batch, which in the original work, in order to avoid plugging in the delivery pipe, had been much too thin. However, as the placer operator acquired greater skill a batch of more desirable consistency could be conveyed without undue trouble from plugging. Also, there was less consumption of air in conveying the thicker charge to the form than in conveying one of thinner consistency.

Wear at Pipe Bends.—Another problem that arose in the first concrete work was the excessive wear at the pipe bends that raised the delivery line from the floor of the drift to the top of the form. At first this elevation was accomplished by the use of a short reverse bend in a single piece of pipe, which often did not survive a pour of 10 cu. yd. Use of two 45° extra heavy steel elbows instead of this reverse pipe bend improved matters somewhat, but the difficulty finally was overcome by the use of 45° Ys instead of elbows. The unused straightaway end of the Y was closed with a threaded pipe plug, which was firmly braced to resist the impact of the concrete. The first charge through the line filled the closed end and provided a protective cushion of concrete

for succeeding charges. The Y also provided easy access for cleaning out obstructions in the bend itself, where plugging was most likely to occur.

Cleaning Pipe.—The 6-in. pipe used for conveying concrete was standard threaded pipe, but it was found advisable to put in flanged unions every third or fourth length in order to facilitate the removal of obstructed sections. It was found that plugging could be caused anywhere in the delivery pipe by too dry a mix, inadequate air pressure, unskilled manipulation of the placer, or the concentration of the large rocks in the aggregate. The latter difficulty made it necessary to limit the maximum size of the aggregate to 2½ inches.

Wear on Pipe.—Experience at Ray indicates that in horizontal straight pipe the wear is almost negligible; the original delivery pipe outlasted the entire group of projects described. The only precautions taken to prolong the useful life of the pipe were to keep the alignment as straight as possible and occasionally to rotate it a few degrees in order to distribute the wear.

Cost

In summarizing the costs of several projects at the Ray mines, $25.24 per yard is found for the first project, $22.00 for the second, and $22.20 for the third. When it is considered that the first project was carried out under a cost-plus contract together with a more expensive type of form construction, while the latter projects were not burdened with this heavy item of overhead, but were planned and executed by employees in the Ray mines organization, the higher cost in the original undertaking as compared with those of subsequent projects is accounted for. For this reason the writer feels warranted in his conclusion that the cost of lining a drift with concrete by the pneumatic method, irrespective of the size of the opening, should approximate $22 per cubic yard. This statement would indicate, as has been the experience at Ray, that, irrespective of the size of the tunnel, the work necessary to prepare it for concrete and the amount of concrete required are more or less in direct ratio when reasonable care is taken in preparatory excavation, so that the thickness of the concrete in place will not greatly exceed the figured requirements.

Advantages of Concrete-lined Drifts

Concrete-lined drifts offer many advantages. They require no repair work, thus eliminating interruption to mine traffic. They satisfactorily take care of the drainage of mine waters. They require little or no track maintenance, experience at Ray in this particular being that practically no maintenance to track was necessary over a period of approximately 10 years with the exception of the replacement of ties on account of natural rot. They provide a watertight overhead protection against the entrance of mine water, which in timbered drifts frequently develops

drip areas, resulting in rapid corrosion of rail and pipe. The consequences of train derailments in concreted tunnels are far less serious than those entailed by similar accidents in timbered tunnels, because the work of getting a train back on the track is never impeded by the presence of timber wreckage, and also because there is no time lost for timber repairs before traffic can be resumed.

Labor

In the concreting of drifts described in this article a large crew of men is not necessary. For the first job it was necessary to import a few key men who had had extensive practical training in underground work involving the use of reinforced concrete and the manipulation of the pneumatic placer. The bulk of the crew was made up of local men, miners and timbermen. All succeeding crews had as their nucleus the local men who had broken in on the first job and they in time came to be a regular arm of the mine organization known as the "Underground Construction Crew," to whom all kinds of concrete work in the mine became almost daily routine.

Acknowledgments

The author wishes to thank Mr. Moses Brown, Assistant Superintendent of Mines of the Ray Division of the Kennecott Copper Corporation, and Mr. Dean La Grange, Chief Engineer, whose help in the preparation of this paper has been material.

Methods and Costs of Handling and Breaking Ore and Rock in Bulldozing Chambers*

By Charles Will Wright,† Member A.I.M.E.

(New York Meeting, February, 1935)

At most mines where large tonnages are handled, "bulldozing" or secondary blasting is an important and costly operation. To reduce the large blocks from primary blasting operations or stoping so that they will be small enough to pass through the chutes into the cars at the haulage level, and not too large for the crushing plant, requires blasting. Often this is done by placing a few sticks of dynamite on top of a block with or without a capping of mud, but to consume less dynamite and to promote safety, most secondary blasting is now done by blockholing, or the drilling of holes 6 to 12 in. long in the blocks, using only a part of a stick of dynamite per hole. Regardless of the method of blasting, however, the term "bulldozing chamber" is applied to the room where this work is done.

Bulldozing chambers permit secondary blasting at all times in a safe place without disturbance to the other mining operations. The chambers are usually high above the haulage level and well below the stope level or open-pit floor. In most instances the material is passed through a grizzly in the bulldozing chamber, but sometimes no grizzlies are used. One to four raises extend from the chamber to the stope floor and one or two from the haulage level to the chamber floor.

Occasionally large slabs of ore or rock choke the raises above the bulldozing chamber, and these have to be blasted by attaching a bundle of dynamite sticks on the end of a long pole and placing them up in the raise between the pieces causing the choke. This is called "chute-blasting."

To give a picture of the progressive developments in the use of the bulldozing chamber, a few typical mines that show the application of this method to both metal and limestone mines have been selected for description here. Limestone mines have been included because many metal miners do not realize to what extent the nonmetallic, or rock, miners are now applying metal-mining methods successfully in their operations. The bulldozing chamber is the "neck of the bottle" for some mines.

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† Chief Engineer, Mining Division, U.S. Bureau of Mines, Washington, D.C.

Alaska Treadwell Mine

My earliest experience with bulldozing methods was at the Alaska Treadwell mine in 1903, when I was assisting Dr. A. C. Spencer in mapping the geology of that mine. Bulldozing was being done both in

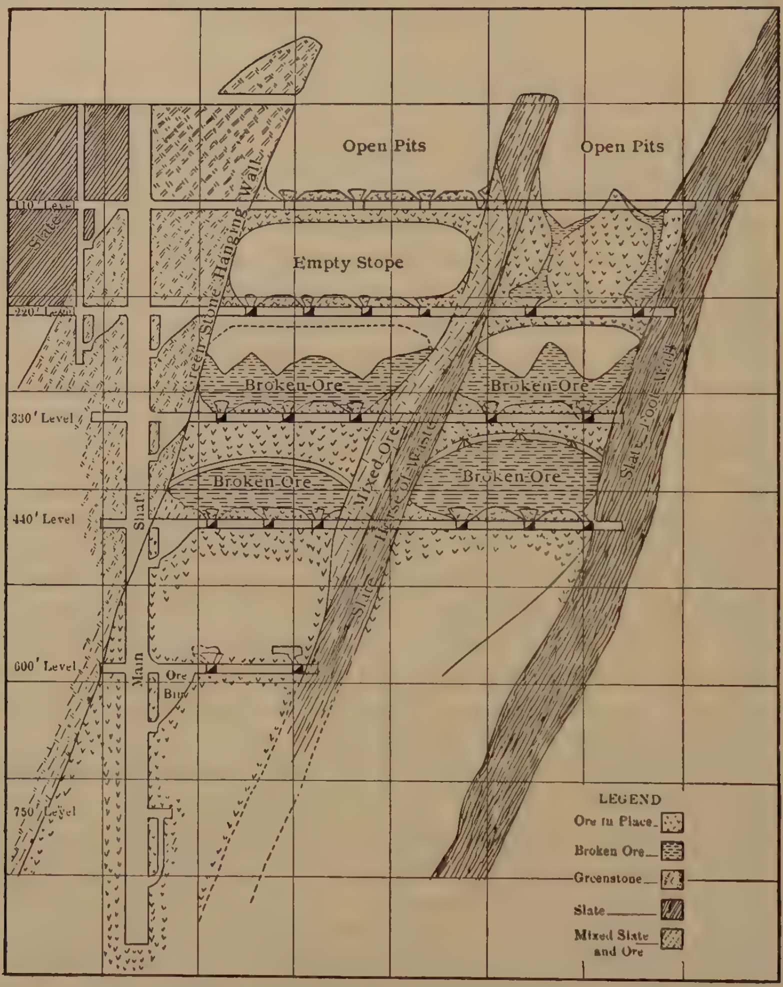

FIG. 1.—IDEAL CROSS-SECTION OF TREADWELL MINE, SHOWING MANNER OF WORKING DEPOSIT (*Kinzie*).

the glory hole and underground. The shrinkage method was used but without bulldozing chambers or grizzlies, and between shifts the large blocks of ore were blasted in the stopes (Fig. 1). Dynamite was used freely, and the consumption was high. I say "freely" because the shift boss who went with us burned sticks of dynamite to illuminate interesting geological details in a stope.

In stoping an average of 33 tons was broken per 10-hr. machine shift, using 0.36 lb. of 40 per cent dynamite per ton. For bulldozing in these stopes 0.85 lb. of 70 per cent dynamite was used per ton mined. This cost of breaking the rock after it had been mined was the largest expense of stoping, for in addition to the bulldozing crew there was a man with a sledgehammer breaking the smaller blocks for each machine driller. However, the record at the Treadwell group for low-cost mining operations, which averaged a dollar a ton, was an outstanding achievement at that time. Robert A. Kinzie, who was assistant superintendent, has described the operations at Treadwell[1].

Alaska Gastineau Mine

One of the earliest applications of the bulldozing chamber was at the Alaska Gastineau mine across the Gastineau Channel from Treadwell. This mine has been described by G. T. Jackson[2].

A combination of shrinkage and caving methods (Fig. 2) was used both in the schist and slate gabbro orebodies, and 45 to 125 tons per machine shift was broken at a cost of 28 and 14¢ per ton, respectively. To this amount must be added the cost of bulldozing, 7½¢ per ton. In stoping, 25 per cent, and in bulldozing, 70 per cent of the cost was for explosives.

In the first attempt made by the operators a bulldozing chamber and grizzly were introduced directly above the ore pockets, but this was unsuccessful because excessive dulldozing made it impossible to hold the timber caps in place (Fig. 3). These were abandoned, and bulldozing then was done from an intermediate drift 20 ft. above the haulage level. In this new arrangement no grizzly bars were used, as most of the blocks could be bulldozed before they got into the chute (Figs. 4 and 5).

Mr. Jackson states that the bulldozers, who must be skilled in the use of powder, work in conjunction with the tramming crews moving from chute to chute to keep them free from rock jams. When the ore is drawn from the chutes the finer material usually comes first, the large rocks working down gradually until they arch over at a point where the chute raise enters the stope, thus preventing any ore from being drawn. The practice then is to tie several sticks of powder with fuse and cap attached to the end of a 1 by 3-in. stick 10 to 20 ft. ong. This charge is set off between two of the large blocks; a large tonnage of ore may be loosened thus and the larger rocks brought down into the chute mouth. It often requires several such charges to bring the blocks far enough down into the chute raise to be reached by a stoping machine for block-

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[1] R. A. Kinzie: *Trans.* A.I.M.E. (1904) **34,** 334.
[2] G. T. Jackson: *Trans.* A.I.M.E. (1920) **63,** 464.

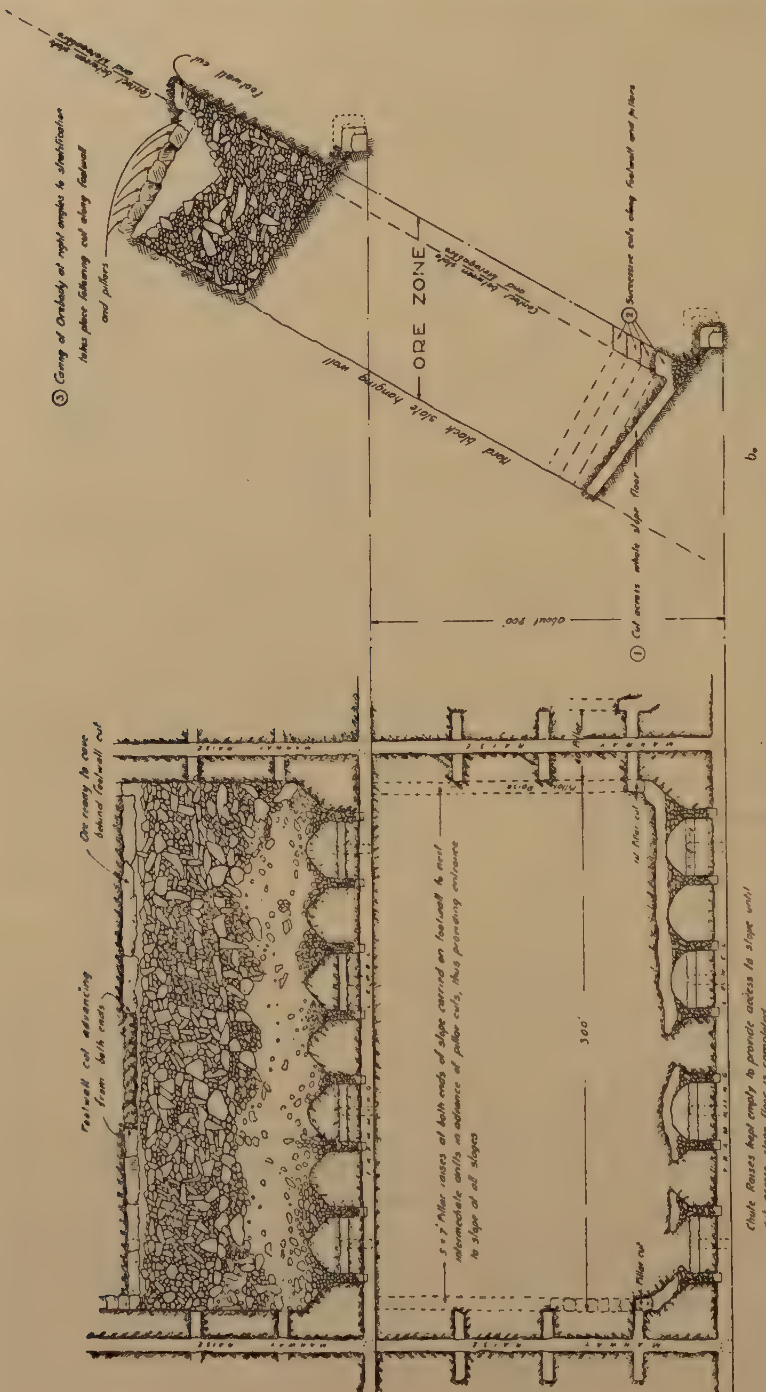


FIG. 2.—COMBINATION OF SHRINKAGE AND CAVING METHOD OF STOPING, ALASKA GASTINEAU (*Jackson*).

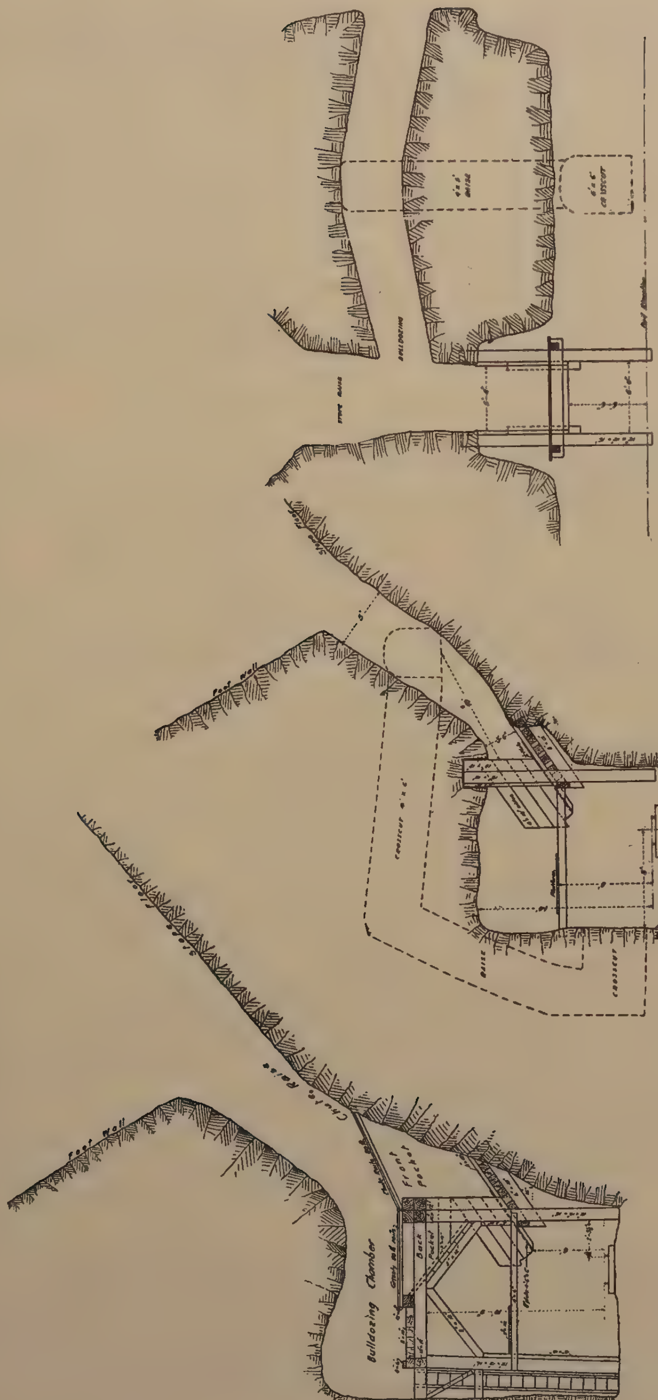


FIG. 3. FIG. 4. FIG. 5.

FIGS. 3–5.—DEVELOPMENT OF BULLDOZING CHAMBER AT ALASKA GASTINEAU MINE.
Fig. 3, cross-section of first chamber; Figs. 4 and 5, cross-section and longitudinal section of modified chamber.

holing. This is the reason for the high consumption of explosives for bulldozing operations, which amounted to twice that for stoping.

Alaska Juneau Mine

Taking advantage of the experience gained at the Treadwell and Alaska Gastineau mines, a wholesale mining method was adopted at the Alaska Juneau mine (Fig. 6), consisting of: (1) cutting out stopes of large area; (2) driving raises from the back of the cut-out stope area through to the next, two levels above; (3) driving numerous powder drifts from and radial to these raises; (4) provision of bulldozing chambers for drawing ore out of the stopes into the loading chutes. Only the last of

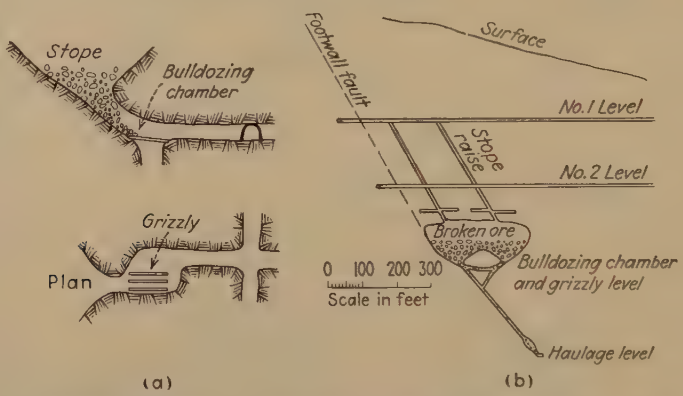


FIG. 6.—STANDARD BULLDOZING CHAMBER (*a*) AND CROSS-SECTION OF STOPE (*b*) AT ALASKA JUNEAU MINE.

these will be discussed. In describing these bulldozing operations Mr. Williams, chief engineer at Alaska Juneau, writes as follows in a recent letter:

In bringing up to date the part relative to bulldozing in the Bureau of Mines I.C. 6186, by Mr. P. R. Bradley, there are only a few changes to be made.

In blockholing, both wet and dry types of jackhammers are now used. For drilling long holes in the large rocks that hang up above the grizzly, the self-rotating machine, wet type, is used with 1⅛-in. hollow hexagon steel.

In blockholing, ⅞-in. steel is used with 1¼-in. gage on finishing steel. For short plug holes short starters are equipped with 1¼-in. bits. Besides the 1¼ by 8-in. Dupont Extra "C" 45 per cent general purpose powder, some 1 by 8-in. 40 per cent Dupont gelatin is used for blockholing. Formerly 40 per cent ammonia powder was used.

Hang-ups formed of large rocks sometimes require 30 or 40 holes. Formerly these were blasted instantaneously with cordeau, but present practice is to use instantaneous electric caps connected in series and fired with a blasting machine. The warning of impending shots is given by sounding an air whistle with which each bulldozing chamber is provided.

The powder ratio for secondary breaking varies with the age of the stope and the nature of the ore. During the first year of operation of a new stope in hard and

tough rock, 3 tons per pound of powder is not an unusual ratio; whereas in the older slate stopes 5.50 tons of rock per pound is ordinary. Secondary breaking on the average yields 4.75 tons per pound of powder.

Of the total powder used in bulldozing (i.e., secondary breaking), 10 per cent is used in mud caps, 25 per cent in drill holes, and 65 per cent in bundles for blasting hang-ups.

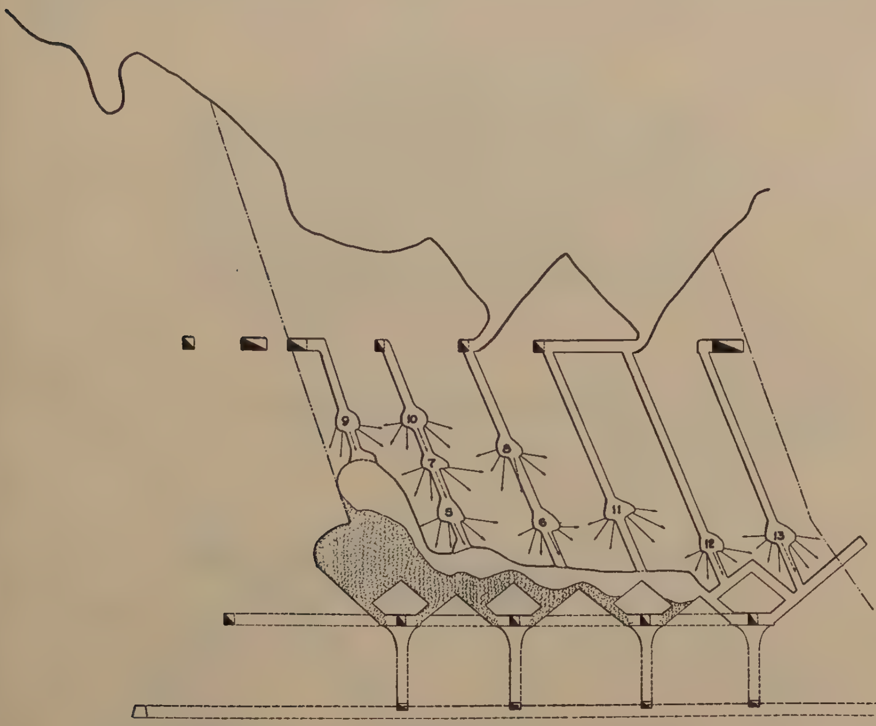


Fig. 7.—North section of 202 stope, Beatson mine, at end of two months mining.

Shows outline of back and rounds to be drilled and blasted during third and fourth months. Numbers on rounds indicate order of blasting.

The grizzly in use for several years is made up of three 16-ft. standard 15-in. @ 108-lb. girder beams. The wide flange is stiffened on both sides of the web by cast-steel fillers spaced at 39-in. centers. A 1 by 15-in. cover plate is riveted to the top, which, besides being a wearing plate, makes the spacing between beams less at the top than at the bottom; this is desirable. The opening between girders is 25 in. The center beam will handle over 100,000 tons of rock before it needs to be replaced.

In primary blasting 20 tons is broken per pound of explosive and 330 tons per man-shift in the stopes. The result for all labor charged to mining was 30 tons per man-shift and total consumption of explosive was 0.4 lb. per ton, of which 13 per cent was for development, 16 per cent for stoping, and 71 per cent for secondary blasting. Total underground

costs are about 30¢ a ton, and 40 per cent of this is for bulldozing operations alone.

BEATSON MINE, LATOUCHE, ALASKA

The system employed at the Beatson mine is much like that at the Alaska Juneau mine. Instead of blasting down the backs of the stopes by large charges placed in powder drifts, long holes were fanned out

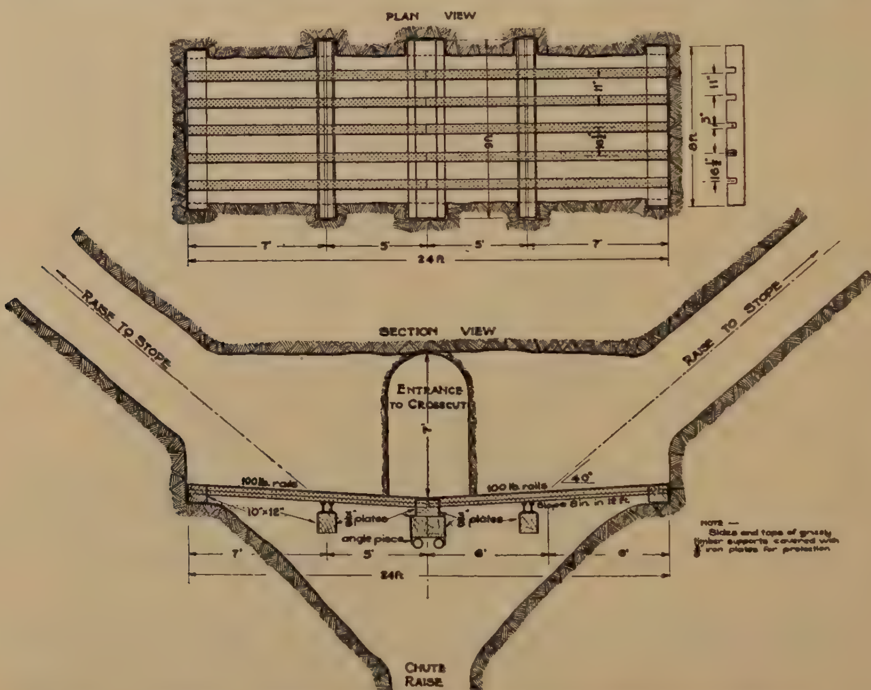


FIG. 8.—GRIZZLY CHAMBER, BEATSON MINE, SHOWING DETAILS OF GRIZZLY.
A third point of draw, not shown in this figure, is installed after the stope is mined. This is done by driving a raise to the stope floor from the side of the grizzly chamber opposite the "entrance to crosscut." This raise lies in a vertical plane at right angles with plane of section shown in sketch.

from benches in raises as shown in Fig. 7. The system has been described by Presley[3], from whose paper the following is quoted:

The tonnage broken per machine shift was 198 tons or 123 tons per stoping shift and the consumption of explosives was 0.245 lb. per ton of ore broken. Total cost for stoping was 15.8¢ and to this must be added 20.8¢ for bulldozing and grizzly repairs. Sixty-five tons were handled per bulldozing shift and the consumption of explosives was 0.16 lb. per ton. The bulldozing chambers were 50 ft. above the haulage level, thus giving storage capacity below the grizzlies. The grizzlies were 24 ft. long, with four or five 100-lb. rails spaced 11 in. apart. . . . The large blocks are either broken with a hammer or blockholed, as no "bulldozing" is allowed,

[3] B. Presley: *Trans.* A.I.M.E. (1928) **76**, 11.

although the term "bulldozing" is used to designate the operation of putting the ore through the grizzlies.

Britannia Mine, Britannia Beach, B.C.

The Britannia method as applied in the East Bluff mine is illustrated in Fig. 9. The block of ore here extends from below the 1200-ft. level

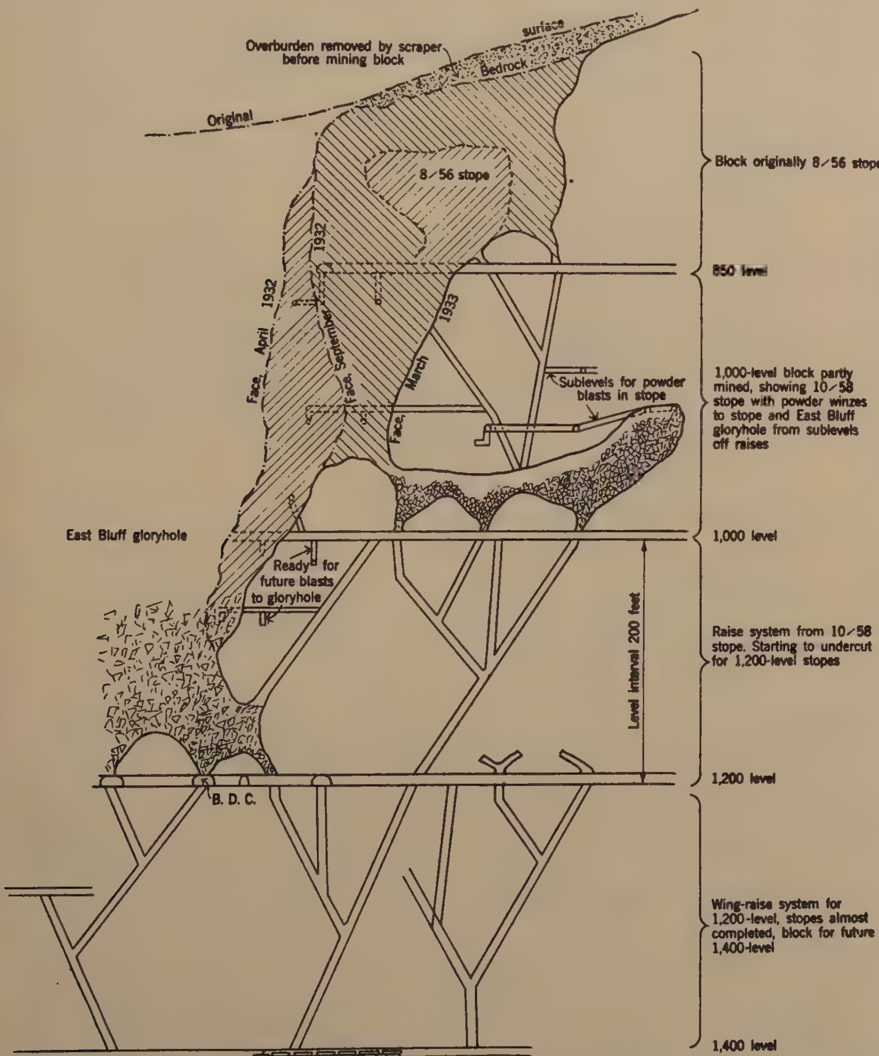


Fig. 9.—Ideal section showing application of Britannia method to East Bluff orebody.

to the surface at about the 700-ft. level and east of the old Bluff glory hole. The intermediate levels are the 850 and the 1000 ft. The block of ore above the 1000-ft. level was first undercut and a series of raises were

made from which, at vertical intervals of about 30 ft., small drifts were driven. In these, powder pockets were made a few feet below the drift level. The explosive for loading these powder pockets was put up in heavy paper bags of 12½ lb. each, and the pockets were covered by a pile of broken material and blasted. Twelve such blasts were required to remove the ore in this block above the 850-ft. level, and the same method is being used to mine the blocks below that level.

The Britannia method has the advantage of being flexible, in that it can be adapted readily to meet changing conditions encountered as mining progresses. No pillars are left to be recovered by auxiliary

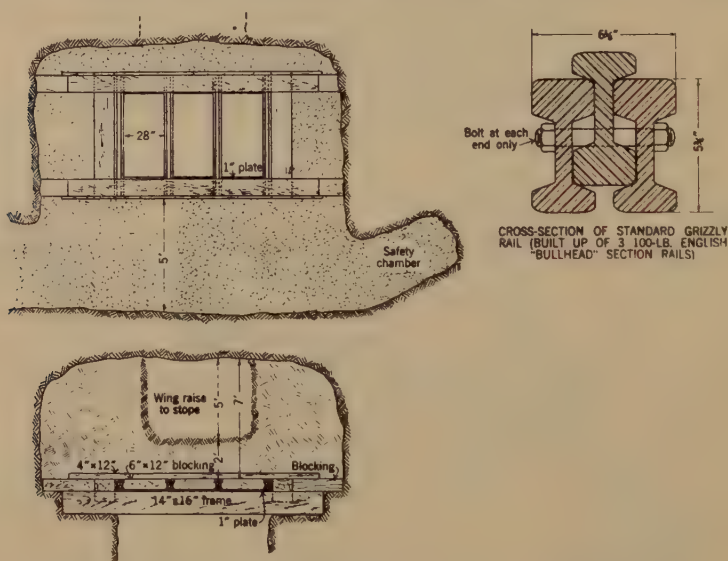


FIG. 10.—PLAN AND SECTION OF STANDARD BULLDOZE CHAMBER.

methods. Mining is constantly retreating into virgin ground. As all working excavations following undercutting are of minimum size the risk of injury from falls of ground is greatly reduced. The mining cost obtained by this method was 23.5¢ per ton, excluding development and tramming.

In describing the bulldozing chambers Mr. Brennan writes as follows:

In carrying out the large-scale stoping operations under the Britannia method the use of bulldoze chambers has been of primary importance. It is here that most of the secondary breaking takes place. A typical bulldoze chamber is illustrated in Figs. 10 and 11.

Grizzly rail construction caused a good deal of trouble in the early stages, and the present method, as illustrated, is the outcome of many experiments, and has been found very satisfactory. The grizzly consists of three or four bars spaced to give clear openings of 24 to 36 inches.

Each bar is built up of three sections of 100 lb. to the yard bullhead rail. The bars lie on cap plates made of 1-in. steel plate, which protect two sides of the 12 by 16-in. fir grizzly timbers.

The short span is important, as the bending moment about the center of the composite bar is thereby kept at a minimum.

The spacing of rails depends on the nature of the rock and on the drop between the grizzly and the transfer chute. The tough East Bluff rock, which has little tendency to shatter during its fall down the raise, is passed through a 26-in. grizzly while the more highly sheared and softer rock in the Fairview section, most of it traveling 500 ft. through the transfer raises, is passed through openings as large as 36 inches.

A bulldoze chamber crew consists of one miner and one mucker. The floor plan is so designed that they are given plenty of room to work in without unduly increasing the amount of floor space that has to be shoveled clear of muck after plugging boulders.

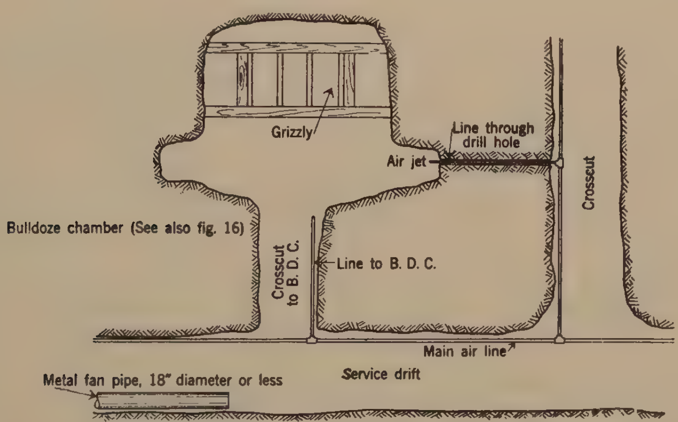


FIG. 11.—PLAN SHOWING VENTILATING SYSTEM IN BULLDOZING CHAMBER.

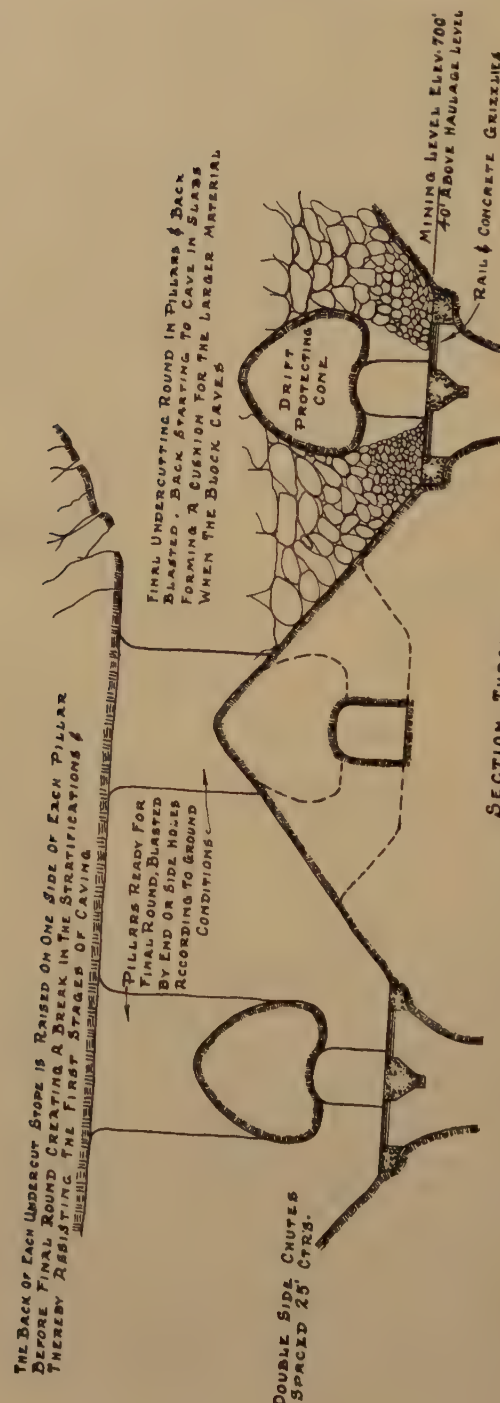
These men are compelled to wear a safety belt, in the back of which a D ring is firmly secured. When the grizzly is open, they must attach to this ring a length of ½-in. rope, the other end of which is attached to an eyebolt in the back of the chamber. The attachment is quickly made at either end by means of a safety snap hook. This hook has been specially designed to carry a heavy load.

The safety chamber, shown in the figure, is cut so that a man may retreat to safety from a rush of muck without stopping to unfasten his rope.

Bulldoze chambers are laid out so as to take best advantage of the ground to be stoped. Usually they are placed 70 ft. center to center, and the footwall row is placed as near as ground conditions will permit to the footwall itself. The raise representing the throat of the chamber then points up the footwall and allows of drawing in that direction to the best advantage.

In blockholing, wet machines are used exclusively. Two types of pluggers are used, depending upon the rock to be broken, one weighing 46 lb. and the other 65 lb. One-inch hollow hexagonal steel is used, the majority of which consists of short pieces resulting from the breakage of longer steel in drifts and stopes.

A self-rotating stoper is available for all bulldoze chambers, these being particularly valuable in this work, as in dangerous corners the drill can be used to put in a hole with the operator at a safe distance away.



SECTION THROUGH UNDERCUT CAVING METHOD NO. 1

FIG. 12, *a*.

Occasionally when very large pieces of rock come down over the chambers it is possible to set up a bar and drill 10 to 15-ft. holes with the regular 4-in. drifting machines. Forty per cent explosive in 1⅛ by 8-in. sticks is used exclusively in the bulldoze chambers. This explosive is not ideal for blasting hang-ups because of its lower velocity but has been found to be the best all-around explosive for use in this work, which consists of blasting hang-ups, drill holes and the occasional mud cap.

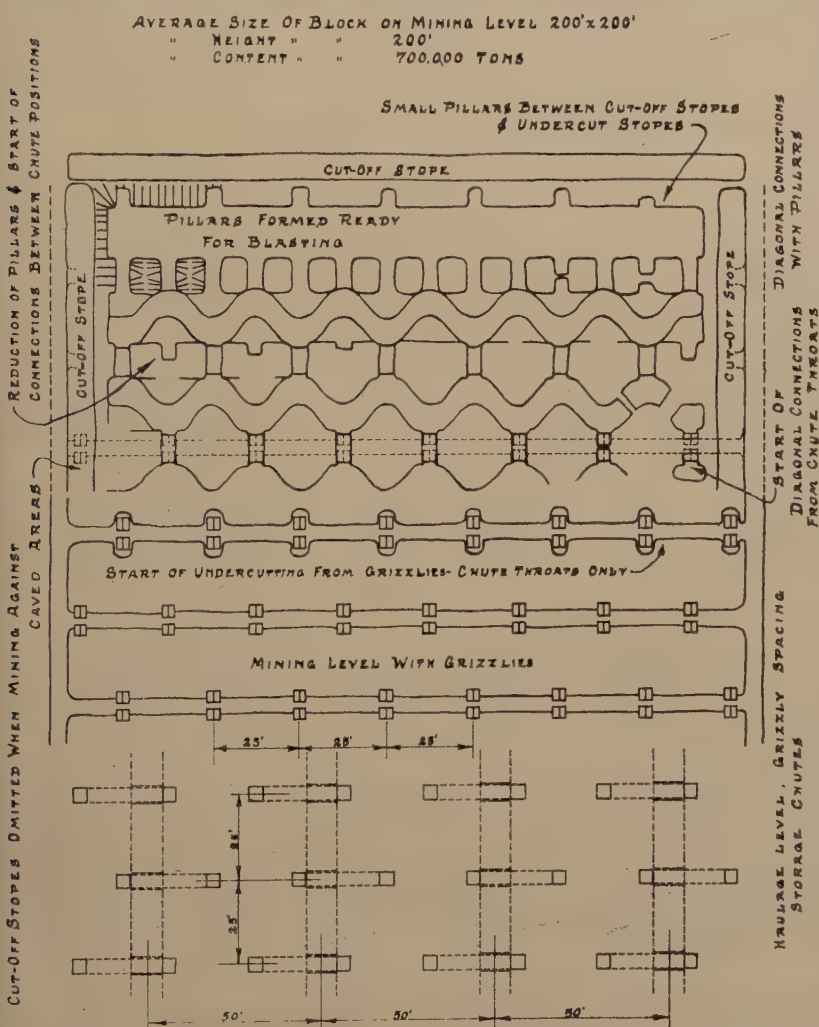


Fig. 12, *b*.

Fig. 12.—Mining limestone by undercut caving method No. 1 at Crestmore mine of Riverside Cement Co., California.

All rock possible is broken by plugger holes but most of the explosive is used in getting the big rocks down into position where it is possible to drill them.

Each bulldoze chamber is provided with an air whistle to give warning before and during all blasting operations.

The labor, explosives, and power used in putting the very hard East Bluff ore through the bulldoze chambers were approximately as follows: Explosives consump-

tion, 0.5 lb. per ton; labor, 0.16 man-hour per ton in bulldozing and 0.11 man-hour per ton, in stoping; power, 0.40 kw-hr. per ton (includes air used in blowing smoke).

Crestmore Mine, California

The caving method used at this mine is an innovation in the mining of limestone. In general, the application is similar to the "sublevel undercut-caving method" used by the Nevada Consolidated Copper Corporation at Ray, Ariz., with modifications to suit the size of the block mined and the class of material (Fig. 12).

C. A. Robotham describes his operations as follows:

> The mining-level drifts running north and south, or at right angles to the haulage drifts below, are spaced 25 ft. apart, with raises driven every 25 ft. connecting with the loading chutes below. These raises are alternately driven vertical and inclined, thus staggering the draw points on the mining level in adjacent drifts.
>
> Undercutting has been developed from a system of undercut stopes and pillars to a room-and-pillar system in which small pillars are left in the undercut areas between the drifts to be reduced in size and blasted out as the undercutting progresses.
>
> The limestone responds readily to this method of mining, breaking well upon the use of properly designed rounds in drifting and raising, shrinking uniformly throughout the entire length of the cut-off stopes whether driven vertically or inclined, and caving according to expectations.
>
> The large material that descends during drawing operations requires no more drilling and blasting to reduce it to the required size than the material of intermediate size which must all be blockholed. A low-strength explosive used in this operation assists in keeping this cost at a minimum.

The bulldozing chambers are 40 ft. above the haulage level, spaced 25 ft. apart, and are 7 by 5 ft. in size. Two raises feed each chamber, the height to the stope floor being 15 ft. The grizzly is of 65-lb. manganese-steel rails with openings 16 by 40 in. Tonnage per hour through grizzly is 9 tons for each of the 64 chambers per block 200 by 200 feet.

Explosive consumption in primary blasting is 0.37 lb. per ton and for secondary blasting 0.15 lb. per ton. Cost for labor, explosive, and supplies per ton through the bulldozing chamber is 13.5¢. Total cost per ton hoisted was 24.8¢. Seventeen tons was mined per man-shift, 0.52 lb. of explosives consumed per ton mined, and 4.3 kw-hr. of power consumed per ton mined.

St. Vicente Quarry, Santa Cruz Portland Cement Co.

This is one of the most interesting developments. Twelve years ago the method used was hand loading and steam shovels, but difficulties arose because the sloughing of high limestone banks brought in overburden, and plans were being made to abandon the quarry and open another more favorable to shovel operations. Robert Kinzie, consulting engineer in San Francisco, who had been mine superintendent at the

Treadwell mines and responsible for the early developments at the Alaska Juneau, conceived the idea of applying a glory-hole method and bulldozing chambers to this deposit. As a result, the costs were reduced to one-third of former steam-shovel operating costs.

An adit was driven into the limestone on a 0.5 per cent grade, two drifts were turned off from this at 60-ft. centers and extended along the principal axis of the deposit. Between the drifts and at an elevation of 30 ft. above the haulage level, a series of bulldozing chambers have been cut in the limestone. Formerly each chamber was 80 ft. long and 50 ft. wide, flat-topped, the bottom being hoppered out and connected

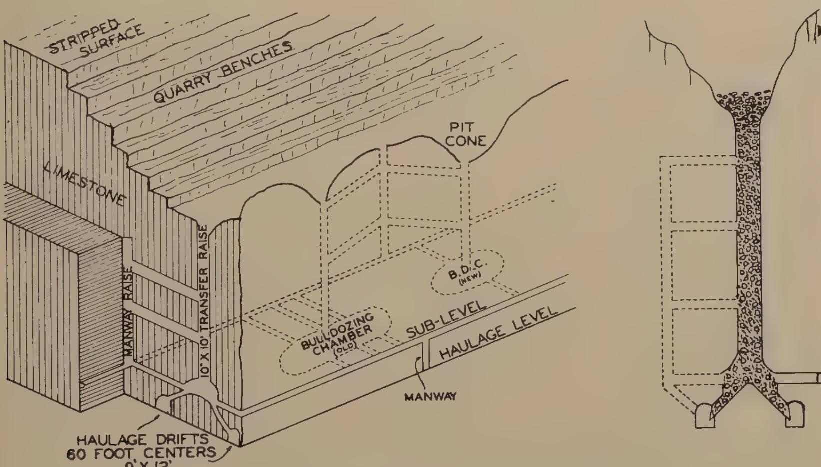


Fig. 13.—Sketch of quarry method, Santa Cruz Portland Cement Co., Davenport, California.

to six chute-raises, three on each side. The chambers are now 40 by 50 ft., with only two chute raises. From the top of each chamber a raise 10 ft. square extends to the floor of the quarry. These raises establish the position of the surface glory holes. They are spaced at intervals of 120 ft. along the major axis. This distance has been increased to 240 ft. for the new raises. Seven raises have been extended to the quarry level at present, and other raises will be extended as they are needed. (Fig. 13.)

Subdrifts are on either side of a line of bulldozing chambers, and crosscuts give access to both sides of the bulldozing chambers. Parallel to the transfer raise is a manway raise with crosscuts to the transfer raise, from which in case of a hang-up it is possible to reach the rock raise quickly through the manway raise and the crosscut next below the hang-up.

In the quarry, blasting is done near the end of the shift, and following this all loose rock is barred down. Bulldozing is done on the succeeding

shift in the pit bottoms. The boulders are drilled and one stick of 40 per cent powder (1-in. cartridge) with 4-ft. fuse is used for the work. Bulldozing underground is done in the bulldozing chambers with 40 per cent powder, whenever a large piece of rock appears. On an average, 6 to 8 rocks require blasting for each 100 tons of rock through the chamber. Long drills are used for drilling, the driller standing on the shelf at the edge of the chamber and drilling the hole with the long steel from this position. As far as possible he avoids any work on the pile of broken rock, except from the vantage points at the sides and ends of the chamber. An eyebolt with a rope attached is placed at each bulldozing crosscut affording additional safety for the men who have to get on the rock pile occasionally. In spite of the bulldozing in the open pits and in the bulldozing chambers an occasional oversize piece gets to the loading chutes, and some bulldozing is necessary at the chute mouth. The final product as loaded into the cars is thus kept well within the limits of the receiving gyratory crusher. The maximum size limit is 60 by 32 in. Signal whistles operated by compressed air are used as warnings of bulldozing at chutes and within the chambers. Safety measures are responsible for the record of no lost-time accident during the past 4½ years.

The cones, transfer raises, bulldozing chambers, and chute raises are kept full of rock. As the chutes are drawn the bulldozing chambers receive fresh accessions of broken rock from the transfer raises and these in turn from the cones and open pits. The subsidence is gradual, and runs of broken rock in the bulldozing chambers are small and always well under control. The generous size of the transfer raise is an important element in obtaining freedom from hang-ups.

Up to 200 tons of limestone an hour is passed through one of these bulldozing chambers, and the total cost of drilling and blasting is 2.78¢ per ton of material. In primary blasting, 0.21 lb. of explosives is used and 0.035 lb. per ton for secondary blasting, or less than ¼ lb. per ton, total consumption; 570 tons is handled per man-shift in the bulldozing chambers, and the total cost per ton is 1.75¢, including explosives and supplies. The total operating cost per ton loaded on cars is 14.3¢. To this must be added the cost of development (0.9¢). Mr. L. R. Davis, quarry superintendent, kindly supplied the above information.

Summary

Of prime importance in mining where bulldozing chambers are used are safety, good ventilation for the workmen, continuity of flow from the stopes to the cars, and low costs, which means high output per man and low consumption of explosives per ton. Large bulldozing chambers and mining methods that use gravity, both to aid in breaking the ore and in delivering it into the cars, are now being applied to large orebodies wherever possible.

A summary of data is given in the accompanying table.

TABLE 1.—*Summary of Data*

	Tread-well Group	Alaska Gastineau	Alaska Juneau	Beatson, LaTouche	Britannia, East Bluff	Riverside, Crestmore	Santa Cruz, St. Vincente
Daily output, tons....	3,000	8,000	12,000	2,265	1,100	5,000	2,000
Mining cost, including development	$1.00	$0.48	$0.297	$0.79	$0.514	$0.248	$0.152
PRODUCTION							
Tons broken per stoping shift...........	18	45 to 125	330	123	720		400
Tons broken per bulldozing shift........		150	125	65	500	72	570
Tons mined per underground shift........	6	16	30	22	23	17.3	70
CONSUMPTION OF EXPLOSIVES PER TON							
	Pounds	Cents	Pounds	Pounds	Pounds	Pounds	Pounds
Primary blasting......	0.36	3.5	0.06	0.24		0.37	0.21
Secondary blasting....	0.85	6.0	0.29	0.16		0.15	0.035
Total blasting........	1.21	9.5	0.34	0.40	0.50	0.52	0.245

Most new mine developments of large deposits are being planned to make more use of bulldozing chambers and the vertical or sublevel stoping system using gravity to break the ore. This applies both to metallic and nonmetallic deposits.

DISCUSSION

(*J. A. Church, Jr., presiding*)

R. K. WARNER,* New Haven, Conn.—I would like to emphasize the fact that this bulldozing chamber method of secondary blasting developed in parallel with the new methods of primary blasting mentioned by Mr. Wright. Also that you cannot afford to use these rather elaborate bulldozing layouts unless you have a great tonnage of ore to pass through each chamber which in turn means an orebody of such a cross section that you can efficiently use these large-scale stoping methods. A corollary, of course, is that the large pieces produced by the large-scale primary blasting makes necessary the facility for secondary blasting provided by the bulldozing chambers. These large-scale primary blasting methods are relatively new and differ from the older forms of stoping. The common element in all of them is the use of heavy charges or at least of the simultaneous firing of large total poundages of powder and as a class they deserve a name of their own such as, for example, "mass" stoping or "roundabout" stoping.

* Mining Engineer.

Drilling and Blasting Practice of the United States Potash Company at Carlsbad, New Mexico

By C. A. Pierce,* Member A.I.M.E.

(San Francisco Meeting, October, 1935; New York Meeting, February, 1936)

Underground operations of the United States Potash Co. at its mine near Carlsbad, N.M., have been continuous since the property was opened about five years ago. Approximately one million tons of potash has been mined and two shafts have been sunk, each to a depth of about 1000 ft. Outstanding in the company's operations is the fact that not a single blasting accident has occurred during the aforesaid period. The purpose of this paper is to describe the blasting practice that has made possible the achievement of this accident-prevention record.

The mining method employed in extracting the thick horizontal bed of potash may be described as a double-entry room-and-pillar system. Mine development is kept well in advance of extraction operations, and a three years' supply of ore, at least, is always available. Limits of the orebody have not been determined, and in the present system of mining headings are advanced by a "shortwall" method. This was the first potash ore bed developed in the United States. The nature of the deposit, therefore, was unknown, and it was deemed expedient to outline a conservative mining policy. Later, when more is known about the deposit and certain control ascertained, a "longwall" panel retreating method may be found advisable.

The necessity for a domestic supply of potash became a national issue previous to and during the World War. The United States Government came to realize that this country should not be dependent upon foreign sources of potash, and began extensive explorations in many parts of the country, seeking an orebody that would meet the exacting requirements of the domestic market. The Department of Interior, through the Geological Survey and Bureau of Mines, made every effort to determine the location of potash deposits and encouraged their exploitation by private capital. Through the combined efforts of the Department of Interior and private industry, the deposit now being worked by the United States Potash Co. was discovered and brought into production.

Manuscript received at the office of the Institute Oct. 4, 1935.

* General Superintendent, U. S. Potash Co., Carlsbad, N. M.

The Deposit

The deposit being developed by the United States Potash Co. is a thick, flat, homogeneous bed of sylvite and halite (a mechanical mixture of potassium and sodium chlorides). The color scheme is decidedly patriotic, the major portions of the bed being mottled red and white sylvite with occasional blue salt crystals. Towards the top of the bed there is usually a pure white band of high-grade sylvite about 2 ft. thick. There is a definite line of parting between the salt floor and the potash bed proper, which consists mainly of a greenish gray clay next to the sylvite bed, followed by salt with a considerable brown clay content, which grades into a relatively poor salt within 10 to 15 in. of the potash bed. The upper limit of the body is not as well defined as the lower limit; the upper 2 to 3 ft. of the bed gradually decreases in potassium content, until it merges into brown clay and salt. The present mining operations do not take this upper 2 or 3 ft. of bed and this portion of No. 4 bed will not be mined until the pillars are being pulled. The present height of headings, from 10 to 12 ft., is as much as is considered advisable to carry under present operating conditions.

The No. 4 bed or seam is normally 8 to 14 ft. thick and is composed of a mechanical mixture of sylvite (potassium chloride) and halite (sodium chloride). Negligible amounts of polyhalite, leonite, kainite, langbeinite, hematite and aluminum compounds (in the form of a greenish gray clay) are encountered. A portion of the bed is coarsely crystalline (1-in. crystals) while in the remainder the crystal growth is of medium size (¼ in.). The potassium chloride crystals vary from clear to milky and bluish white; also from reddish brown to red. The reddish color is due to minute particles of iron oxides.

Generally speaking, the deposit is very regular in potash content, gradient and thickness. Occasionally undulations occur in the contact between salt and potash—the floor of the deposit is undulating to a minor degree—but to date these irregularities constitute less than 0.5 per cent of the area developed. Occasional salt horses are found. They have been classified as depositional, erosional and structural, and are being studied intensively.[1]

Electrical Equipment

Since there was no precedent to go by in so far as mining procedure was concerned in this country, much time and study were required to develop a mining method applicable to this deposit; such a mining method necessarily included a safe and practical system of electrical blasting.

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[1] See R. V. Ageton: Salt Occurrences in the Potash Mines of New Mexico. A.I.M.E. *Tech. Pub.* 686 (1936).

Four transformer stations (2300 to 220 volts) are installed in the mine, from which current is transmitted to the working faces for power purposes, lighting and blasting. Separate wires are used for blasting, the maximum length of the blasting lines being 3000 ft. Two magnetic contactor switches (60 amp., 220 volt, two pole) are in use at each transformer station. The final connection in the blasting circuits is made with these switches; the other side of each switch is connected to a safety switch, also installed on the transformer panel and this safety switch, in turn, is hooked to the 220-volt power circuit of the transformers. Magnet coils of the contactor switches are connected to switches at No. 1 shaft station, whence all blasting is done after the blasting crew has been checked out of the mine.

This electrical system is equipped with a number of safeguards: The main blasting switches, near the shaft, are kept locked except when firing; the safety switch at the transformer station is locked "open" after blasting by the oncoming shift; it remains so locked until the faces are cleared at the beginning of a shift, and it is always checked to see that it is open before a blaster does any loading of holes at the face.

Drilling and Loading Holes

Potash as found in this mine is difficult to break with an advancing face. The ore is not hard, but it is tough and tenacious. When blasted,

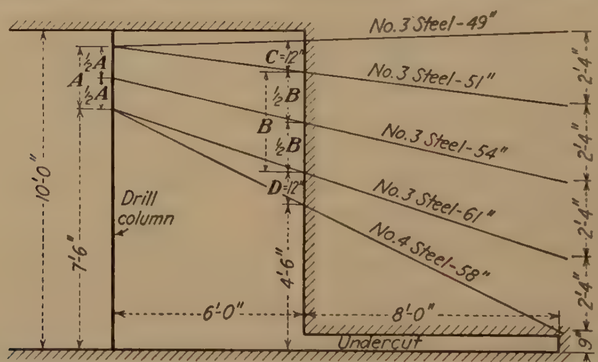

Fig. 1.—Standard round for faces from 9 to 12 ft. high, U.S. Potash Company mine.

the ore tends to break in slabs rather than fine pieces. Owing to a marked decrease in mechanical loading efficiency in mucking a face containing large slabs, breaking the ore into small lumps has been found advantageous.

As an aid in blasting, all faces are undercut by a short-wall undercutter to a depth of 8 ft. There are several advantages in undercutting this kind of deposit. A material saving in explosive cost is obtained. A smooth surface upon which the mechanical loading units operate increases the efficiency of these units. Under certain conditions it is probable that blasting the entire face without the use of undercutters and loading "off

of the rough" might show a lower per ton cost. Such instances however would probably be exceptions rather than the rule. The rate of cutting is 4 to 6 in. per minute advance using an 8½-ft. cutter bar; the rate of cutting 100 sq. ft. is 30 minutes.

The breast holes are drilled with mounted electric auger drills, powered with a 1½-hp. motor. Holes are 1⅝ to 1⅞ in. in diameter, and

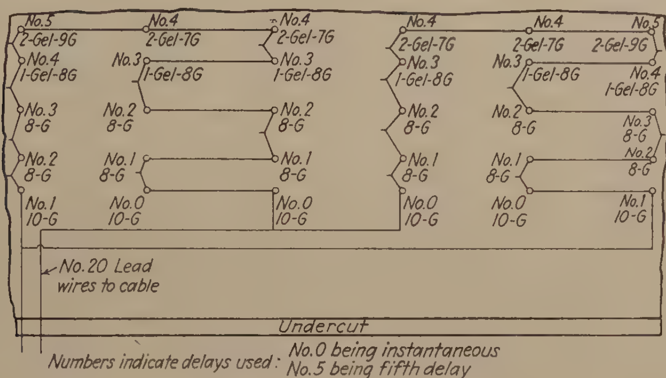

Fig. 2.—Delays used in blasting, typical 20-foot face.

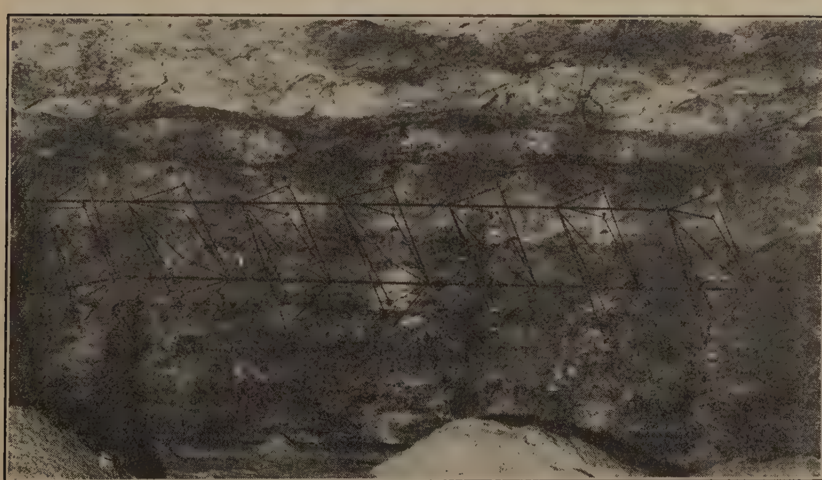

Fig. 3.—Hook-up used in firing from blasting circuit.

have an average depth of 8½ ft. A special detachable bit is used; the starter bit is three-point, high center, the second is four-point, straight-line. The rate of drilling is 32 in. per minute.

A standard round has been developed (Figs. 1 to 4). Although this round is satisfactory, we believe that it can be improved. Experiments are being made with different rounds, explosives and firing patterns.

All loading and blasting of faces is done at 5:00 a.m., on a shift when the mine is not producing ore. The blasting is in charge of a specially

trained blasting crew that consists of one boss and from one to three men, depending on the number of faces to be blasted. By keeping highly trained men on this particular job, missed holes are reduced to a minimum (1 in 10,000 shots) and many hazards caused by lack of experience are avoided. A heading having missed fire is isolated and the next blasting crew fires the missed hole.

Dupont Gelex 2 (43 per cent strength) for primers and Extra G (25 per cent strength) dynamite with delay electric blasting caps are used with satisfactory results. Our blasting practice necessitates delay caps Nos. 0 to 7 inclusive. Leg wires are 12 ft. long. In loading the holes of the standard round employed, the primer or stick of powder containing

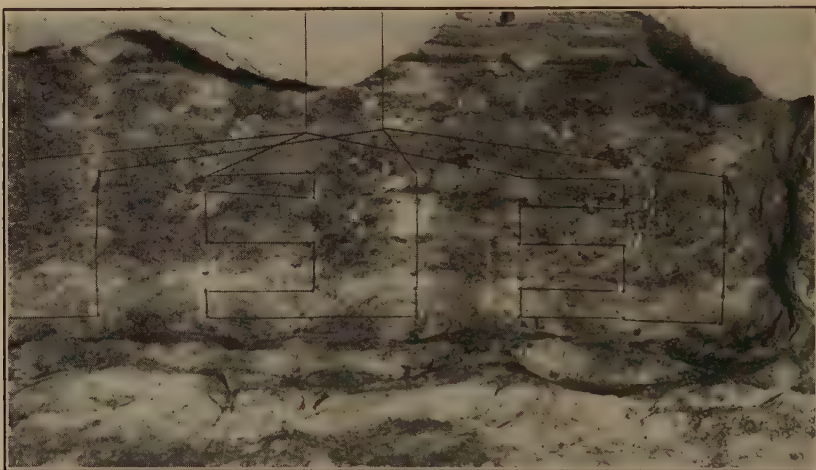

Fig. 4.—Hook-up used in firing from storage battery.
This is left half of typical room heading, 40 ft. wide by 12 ft. high.

the detonating cap is the second stick from the bottom of the hole. On an average, nine sticks per hole are used, and each stick is split so that the entire charge may be well tamped. The consumption of dynamite per ton of ore is *less than* one pound.

A straight parallel hookup is used on ordinary rounds, but, owing to an excessive number of holes in one blasting circuit, a parallel-series type of connection must be used at times. No. 18 cotton-covered wire is employed for the connections between the leg wires of the caps and the permanent blasting circuit; this distance is never more than 20 or 30 ft. The blasting circuit from the face to the transformer stations is No. 8 rubber-covered wire, and the circuits from the transformer stations to the shaft are No. 14 rubber-covered wire.

Magazines

All dynamite is stored on the surface in a reinforced concrete magazine, which is about ¾ mile from the mine yard and is well ventilated

and protected by a bullet-proof steel door, set in concrete. This door is kept locked at all times. The magazine where caps are stored is

FIG. 5.—UNDERGROUND DYNAMITE MAGAZINE (ONE DAY'S STORAGE ONLY).

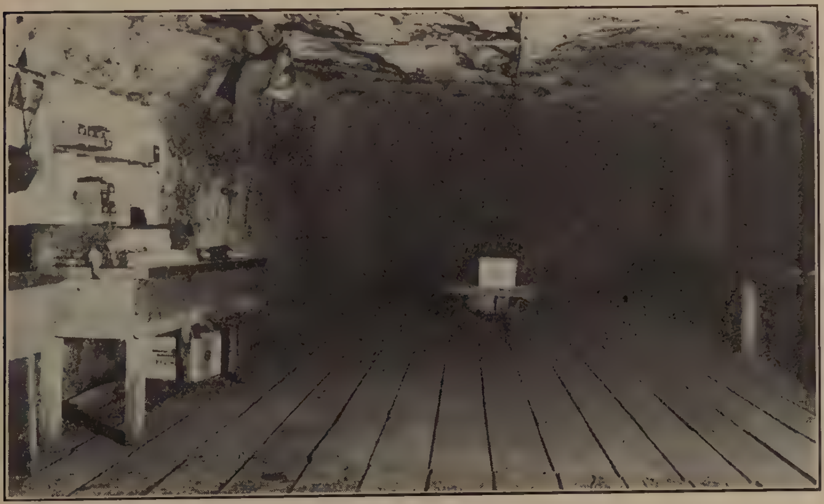

FIG. 6.—UNDERGROUND PRIMER MAGAZINE (ONE DAY'S STORAGE ONLY). Shows arrangement for making up primers (one day's supply); also the entrance to the long, narrow entryway connecting the dynamite and primer magazines, which is closed to traffic.

100 yd. from the dynamite magazine, and of similar construction. The underground powder magazine (Fig. 5) is in a dead heading, a safe distance from the shaft and the working faces. It is roomy and has convenient tables on which to store the powder All electric wiring is in special

conduits, and lights are of the explosion-proof type. The magazines are kept locked except when being used.

Construction of the primer magazine is similar to that of the dynamite magazine. Primers are all made on the day shift by one man. The primer magazine is parallel to the dynamite magazine. The two magazines, 100 ft. apart, are connected, for ventilation, by a 4-ft. crosscut, which is closed to traffic by an immovable grating. A bench on one side of the primer magazine provides a suitable place for making up the primers, and on the opposite rib the finished primers are stored in boxes in which the various delays are segregated. (Fig. 6.) The entrance is screened with wire of extra heavy mesh and the door is padlocked.

In making up the primers, a hole the size and length of the particular delay to be used is made in one end of the stick of dynamite by means of a wooden skewer. The cap is then inserted in the hole and forced well out of sight by pressing the stick of powder against a rubber-covered block. A half-hitch is then taken around the stick of powder with the leg wires, to relieve the cap of any strain that may come on the wires. The remainder of the wire is wrapped around the stick of powder. The leg wires of the cap are shunted at the factory, and this shunt remains intact until the hook-up at the face is made. There are several methods of shunting leg wires; all methods firmly hold the two wires together by some form of metal fastening.

Transportation of Dynamite

Enough powder for 24 hr. is transported by truck daily from the surface magazine to the shaft collar. It is then lowered through the shaft to the working level with as little delay as possible. It is moved in a specially built powder car by a battery locomotive, from the shaft station to the magazine or the faces.

The powder car is a strong oak box, with a capacity of 24 boxes of dynamite, mounted on a roller-bearing mine-car truck. All bolt heads and nuts in the box are countersunk and sealed off with insulating rubber compound material. Each side of the body is made up of loose boards that fit in slots so that they may be taken out one at a time. Each topmost side board has a shoulder that fits over the top of the car, making a tight seal when all side boards are in place.

Blasting Procedure

Information as to the faces to be blasted and the amount of powder required is left for the blasting crew by the preceding shift. At the beginning of the blasting shift, the crew transports the powder from the surface magazine to the faces. On their first trip past the transformer stations, the blasters check the safety switches, to be sure that they are padlocked open. When distribution of the dynamite is completed, the blasting

crew obtains the necessary number of primers at the primer magazine and proceeds to the first face to be loaded.

The safety switch near the face is checked, so that the crew is positive that it is open. The crew then loads the face and makes the various wire connections, with the exception of the connection to the blasting circuit, which is left open until the power is turned off in the entire mine. The blasting crew then proceeds to the other faces, and the same procedure is followed until all faces are loaded and ready to blast.

At the end of the shift, and before the blasting crew enters the mine, the blasting boss goes to the main distribution transformer station near the shaft at the surface and pulls all power switches serving the mine, breaking all circuits throughout the mine. He then notifies his crew that the power is off, and the men proceed to the faces again, where they make the final connection in the blasting circuit. The safety switch near each face is closed, and the crew proceeds to the transformer station that serves that particular group of faces. The safety switch on the transformer panel is unlocked and closed, establishing a complete circuit from the loaded faces to the power circuit, except the magnetic contactor switch, which is still open. This procedure is followed in all groups of faces at their respective transformer stations.

When the blasting circuits have been completed, the blasting crew proceeds to the shaft, where all men are checked out of the mine, and the power is turned on at the main transformer station. The blasting boss then unlocks the boxes that contain the main blasting switches. Each switch, controlling a section of blasting circuit to be energized, is closed and opened immediately, actuating the magnetic contactor switch to which it is connected, which, in turn, energizes its blasting circuit and fires the loaded rounds.

This method of blasting is very satisfactory if operations are within a certain radius from the shaft. When extraction is so far advanced that the cost of blasting lines is not commensurate with results obtained, an entirely new set-up will be necessary.

We have anticipated such a condition and have developed a method of blasting with storage-battery locomotives, using definite safety precautions (Fig. 4). This new method will be inaugurated when conditions necessitate.

Influence of Rock Structure on Blasting

By William B. Plank* and Albert H. Fay,* Members A.I.M.E.

(New York Meeting, February, 1935)

In practically all rock-excavation problems there is need for a careful study of the rock structure, its fault, cleavage or bedding planes, and even the texture of the rock itself. These studies should form the bases upon which an excavation method is selected and they should be carefully considered in predicting the amount and extent of the excavation.

The need for such studies was forcibly illustrated in the problems involved in a cut made in 1929, along and underneath a 200-ft. high rock projection for a State highway near the City of Easton, Pennsylvania. The rock mass was badly faulted and also contained a number of cleavage planes at approximately right angles to the fault planes. Much of this structure was visible and could have been studied and analyzed before the work was begun. The State had rigid specifications as to finished grade and slope of cut, which apparently were made without reference to the geologic structure. The contractor complied with the specifications, under the close inspection of the State engineers. A tunnel blast was placed in the base of the cliff, with the result that 65 per cent more rock was broken than was intended by the specifications.

About four years after the blast, the writers of this paper studied the circumstances surrounding the operation, to determine the cause of this excess excavation.

Scope of Work

The section of the roadway involved was about 200 ft. long and on a 6° curve. It was necessary to relocate points on the curve, from which all faults and cleavage planes were located and measured by the use of transit and clinometer.

A plan (Fig. 1) showing the blasting tunnel, the roadway, and traces of the faults and cleavage planes on the level of the roadway, was prepared; also 13 cross-sections at varying intervals, showing the original face of the hill and the face after the blast. The original and final profiles of the rock face, taken from the plans of the State engineers, are shown in Figs. 2, 3 and 4.

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* Department of Mining Engineering and Metallurgy, Lafayette College, Easton, Pa.

The drawings, while legible to an engineer, did not present the case clearly to a layman; so a model, showing both plan and sections, was

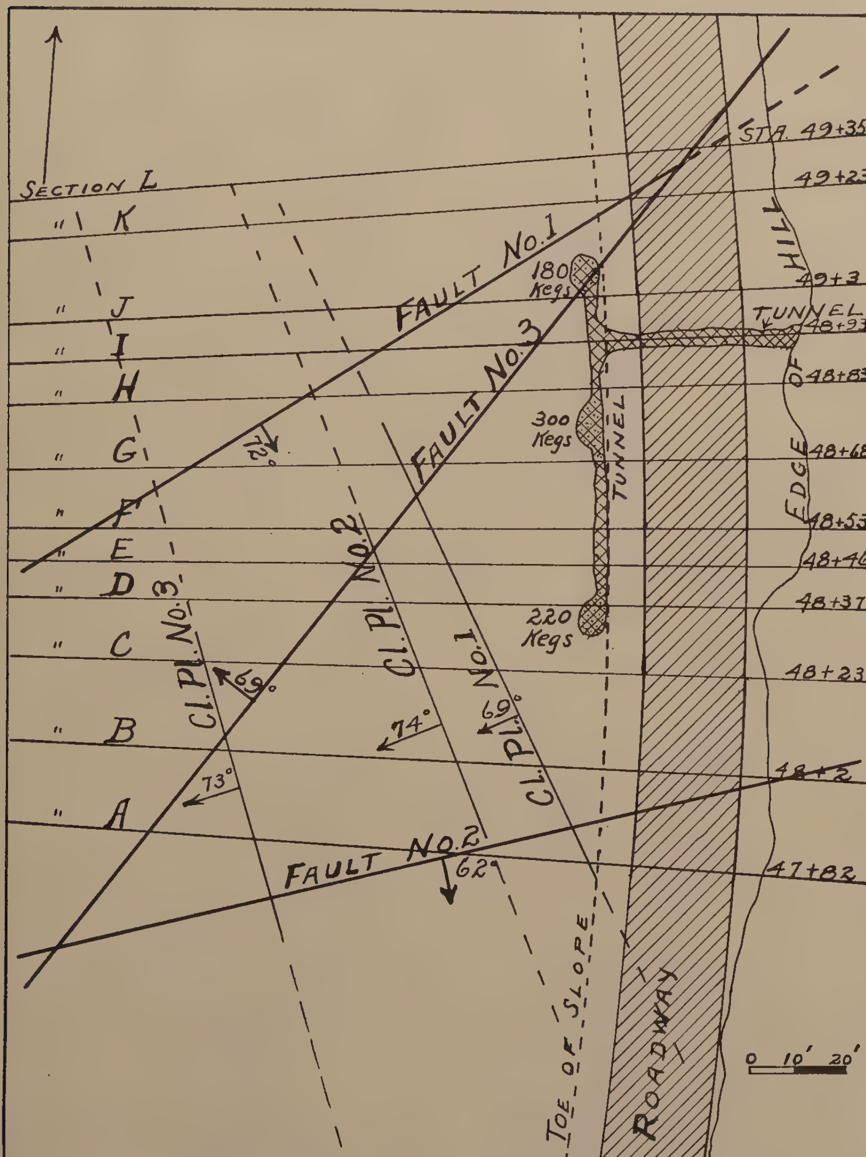

FIG. 1.—PLAN OF FAULTS AND CLEAVAGE PLANES ON ROAD LEVEL.

constructed of glass, by tracing each section on 8 by 10-in. glass sheets, using quick-drying colored enamel paints, on a scale of 1 in. = 20 ft. When the tracing was dried, a cover glass of the same size was placed

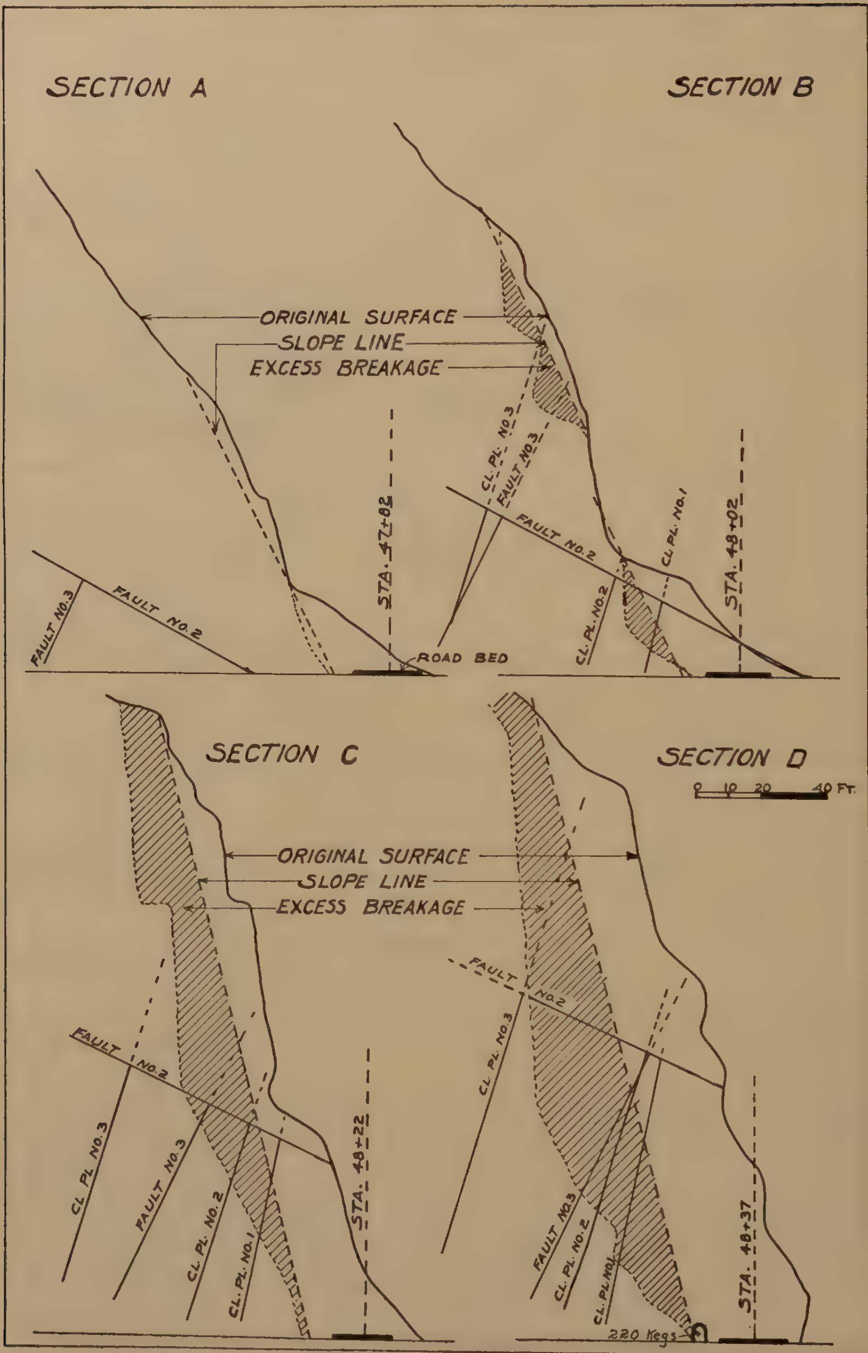

Fig. 2.—Profiles of rock face.

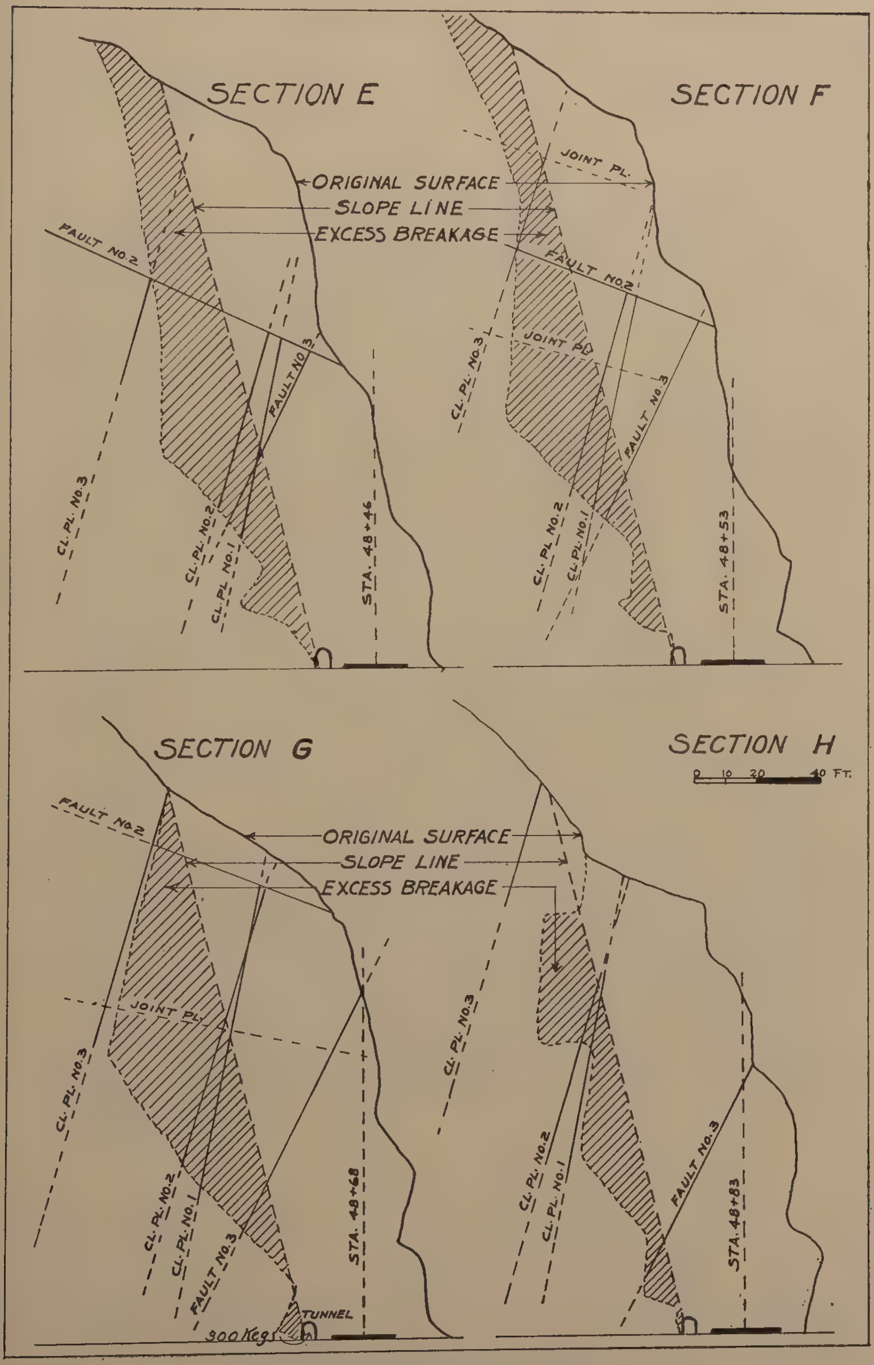

FIG. 3.—PROFILES OF ROCK FACE.

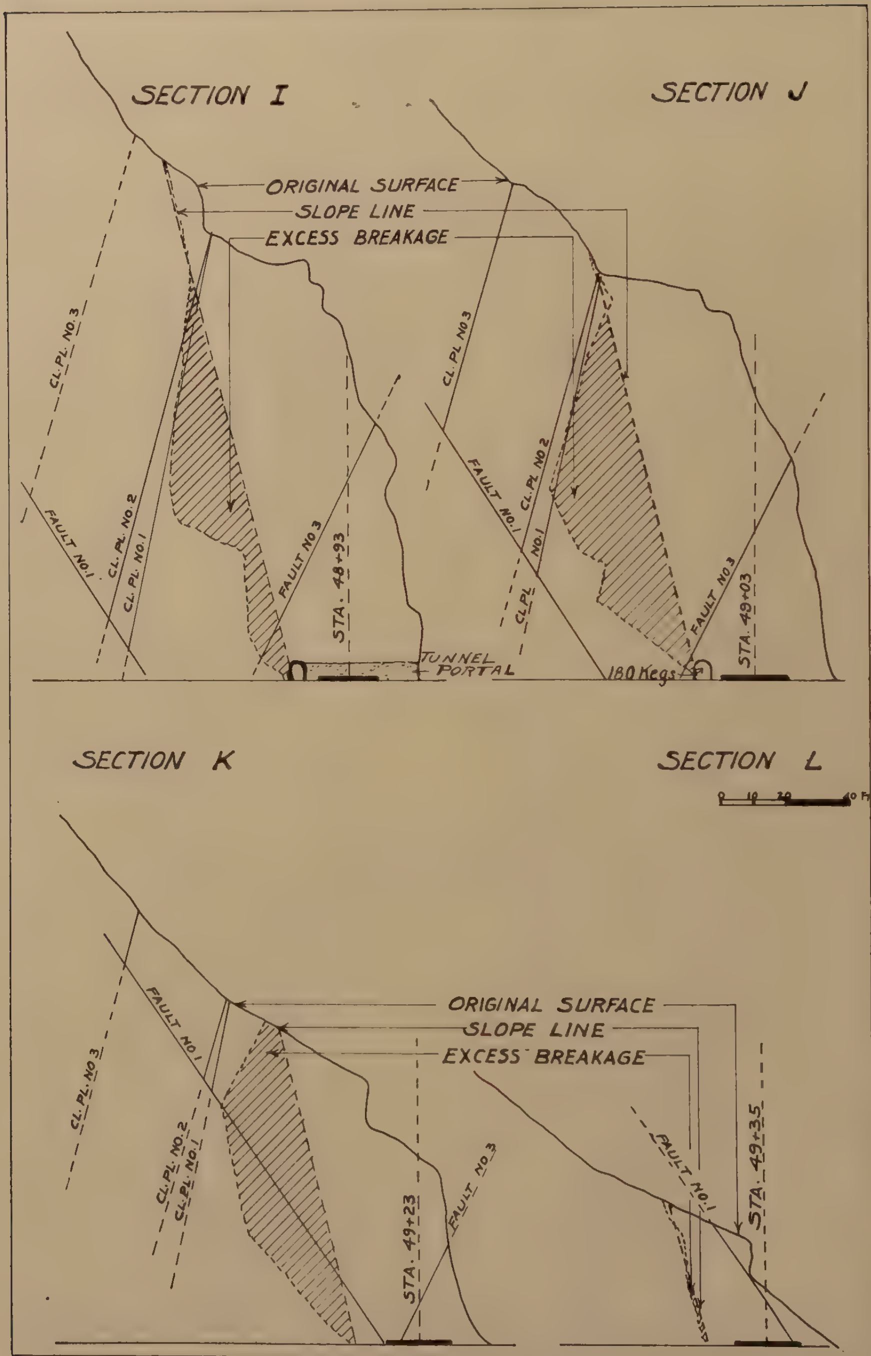

FIG. 4.—PROFILES OF ROCK FACE.

upon it and the edges were bound with tape. The glass sheets were then mounted vertically in a frame, superimposed upon a plan to the same scale, and arranged in their proper radial positions. This method of presentation clearly showed the various rock features in their true relation, and it proved an effective aid to the laymen who considered the

Fig. 5.—Final rock face.

case. The model (Fig. 6) is in the museum of the Department of Mining Engineering and Metallurgy at Lafayette College.

On a photograph of the final rock face (Fig. 5) are marked the principal cleavage or joint planes that are shown on the drawings and glass model.

A motion picture, taken by the contractor, at the time of the blast, reveals in clear detail the movement of the rock face and the time elements involved.

The Rock

The cliff, as described by George H. Ashley, State Geologist of Pennsylvania, consists of gneiss and schist of Archean age, highly contorted, intruded by pegmatite dikes, and full of cleavage and joint planes, and slickensided planes, all the result of repeated crustal movement. The rock is hard and fine grained. An examination of the rock structure

Fig. 6.—Glass model.

showed three major planes of weakness, indicated on the plan and sections as faults 1, 2 and 3. These faults stand at high angles, each shows movement, and their strike runs into the hill at angles of from 40° to 70° with the center line on the roadway. They intersect each other within the excavated area, along a line 125 to 150 ft. above the roadway.

Running at approximately right angles to these fault planes, and practically parallel with each other, are a number of highly inclined cleavage or joint planes, the principal ones of which are denoted on the plans as cleavage planes 1, 2 and 3. These planes dip into the hill at angles of from 69° to 79° to the horizontal. Another system of planes, marked joint planes on the sections (*F* and *G*, Fig. 3), stand approximately at right angles to the cleavage planes and dip towards the river. That the rock structure was weak and unstable was shown plainly by this system of intersecting planes.

Data on the Blast

The method of blasting was that known as "coyote" or "tunnel blasting." The tunnel, about 80 ft. long, was located at and along the ditch line of the finished roadway. It was entered by a portal about 40 ft. long at right angles to the roadway.[1]

The tunnel was loaded with black blasting powder, concentrated at three points. The quantities involved in the blast were as follows:

Amount of explosives............	700 kegs each of 25 lb. = 17,500 lb.
	Cu. Yd.
Calculated rock to be excavated to theoretical slope line, ¼ to 1.	17,496
Total rock removed by blast..................................	28,900
Excess rock broken..	11,404
Rock later taken from river................................	7,252

Choice of Method

In choosing the blasting method, the following criteria appear to have been pertinent:

Application of Tunnel Blasting.—In this kind of formation the authorities recommend the method of blasting used in this case.[2]

This method is decidedly not applicable to horizontally bedded formations, because of the danger from overhangs. The principle involved is the blowing out of the toe or bottom of the cut, so that the top will fall by its own weight and will not be thrown a great distance from the face. Undoubtedly the contractor took all of these features into account.

Location and Size of Tunnel.—The height of the face in tunnel blasting should at least equal the length of the tunnel running into the hill, and this method should not be used for cuts less than 60 ft. high. An examination of the contractor's records shows that the length of the

[1] A good description of the tunneling operations is printed in *Du Pont Magazine* (May, 1930) 10.

[2] S. R. Russel: *Du Pont Explosive Service Bull.* (Jan., 1930).

Cushing and Peterson: Modern Blasting in Quarries and Open Pits. Hercules Powder Co., 1927.

portal to the tunnel was about 40 ft. and the cover over the entire length of the tunnel averaged over 100 ft. It appears therefore that the tunnel was designed properly.

Loading of Explosives.—The recommended loading of explosives per cubic yard of rock to be moved for black powder is from 1 to 1½ lb. In this case 17,500 lb. of black powder was used for a calculated rock excavation of 17,496 cu. yd.; or 1 lb. of powder to 1 cu. yd. of rock. The contractor followed conservative practice in calculating the amount of explosive.

As to the use of black powder versus dynamite for this blast, there seems to be not much choice between the two.

Comparison of Tunnel Blasting with Other Methods.—Another method of blasting that could have been used is known as the "bench" method, whereby the excavation is made from the top downward by drilling and blasting successive series of vertical holes; say 16 to 20 ft. deep.

In the first place, where there are highly inclined seams in the formation, as in this instance, there is great difficulty in drilling because the drill steel tends to follow along the seams. Furthermore, in blasting, higher charges would be needed and the force of the explosives would follow along the seams and tend to shatter the whole face of the rock cut. In this instance the effect upon the rock face would undoubtedly have been more severe than with the method actually used. Furthermore, the writers believe that slides of triangular blocks of rock made by the intersecting fault and joint planes would have occurred also by this method. These slides might not have taken place at once after blasting, but undoubtedly would have occurred eventually, and the subsequent work at lower levels would have been extremely hazardous.

Theory

As mentioned, the principle behind the tunnel method of blasting is that the toe or bottom of the hill is moved out by the explosive and the top falls along the natural slope line. Where the rock is horizontally bedded, there is danger of the occurrence of an overhang at the top, but where the formation is steeply inclined, as in this case, the final rock face will be inclined according to the system of planes that exist in it.

The rock blown out at the toe, in theory, is wedge shaped, the base of the wedge being the original rock face and its altitude being the "line of least resistance," or the shortest distance from the original face to the explosive charge. In this case the apex of the wedge can be considered as corresponding with the length of the crosscut part of the tunnel, and the altitude would then be equal in length to the portal part of the tunnel. The volume of the wedge depends upon the hardness of the rock and the amount of charge, but nowhere is the angle at the apex greater than 90°.

Remembering then that the bottom of the bank would be the first to move, because it would be propelled by the direct action of the explosive, there should have been a small interval of time before the top of the bank began to fall. From the evidence on this point, as revealed by the motion picture film taken at the time of the blast, there was a decided interval of time between the moving out of the toe and the falling of the top. The motion picture reveals also that after an interval of about ½ min. there was a secondary movement of rock from the top. It may be assumed, therefore, that the results of this blast were strictly in accordance with theory and that the action of the explosives was confined to the bottom of the cut.

Effect of Seams on Action of Explosives

Where the explosive is in contact with seams in the rock, it is to be expected that its force would follow along these seams, which in that case become lines of least resistance, and a blown-out shot would result. This force would naturally tend to find its easiest way to the surface, and the plans show that if this had happened in this blast the force would probably have reached the surface principally along fault No. 3 well within the ground that was calculated to be excavated. (See sections E to H.) Fault No. 2, which intersects fault No. 3, does so along a line that is much farther from the explosive charge than the line of least resistance, and was at no place in direct contact with the explosive charge. Therefore it appears improbable that the force of the explosive followed far along the fault and joint planes in a vertical direction. If it had, the whole face would have moved out almost simultaneously. This apparently it did not do, as shown by the motion picture film.

From all of the evidence revealed by the investigation, the writers conclude that after the moving out of the toe of the cut, which may have included the wedge-shaped mass below faults 2 and 3, the material above these faults, because of the system of intersecting joint planes running through it, slid out by its own weight until the angle of repose was reached.

Conclusions

The authors concluded that:

1. The proper method of blasting was used, the charges were properly placed and, if anything, less explosive was used than is customary for this kind of blast.

2. The bench method of excavation would probably have resulted in as much, if not more, excess rock slide, because of the direct contact of the explosives with the entire rock face. This method would also have been extremely hazardous to the workmen working at lower levels.

3. The effect of the blast as observed by the camera was normal, the toe moving out first and the top falling in.

4. The force of the explosives probably did not follow far along the seams in the rock in a vertical direction.

5. The excess fall of rock can be attributed to the instability of the material, due to the system of faults and joint planes running through it, which caused slides beyond the theoretical slope line until the angle of repose was attained.

Summary

The construction of a highway around Paxinosa Point, on the Delaware River near Easton, Pa., made necessary the blasting and removal of a high rock cliff. The rock was shattered by means of a tunnel blast, which brought down a much greater amount of rock than had been calculated. The writers undertook a study of all the circumstances surrounding the blasting operation, to determine the cause of the excess rock movement. They reached the conclusion that neither overloading nor blasting method was at fault, but that the excess rock slid down of its own weight, owing to the presence of steeply inclined fault planes and joints.

The results of the blast and the rock features were illustrated by drawings, and particularly by a glass model, for the benefit of the parties to a claim for payment for subsequent removal of part of the excess breakage. As a result of this presentation the contracting company that engineered the blast was successful in its claim.

DISCUSSION

(*C. W. Wright presiding*)

G. H. Ashley,* Harrisburg, Pa. (written discussion).—In reference to the conclusion that "the proper method of blasting was used," it may be noted that (1) almost twice as much rock was displaced as specified, of which 7,252 cu. yd. had to be moved from the river at large expense; (2) the figures given do not include an additional large amount of rock broken but not removed and now forming a weak talus slope against the foot of the cliff, rather than a supporting base, and (3) the cliff face as left is much more precipitous than originally planned, and, I am told by the Highway engineers, has required the removal of an additional 1000 cu. yd. for safety, with the prospect of having to remove many parts of the cliff in the interest of future safety.

Granting that the bench method of removal would have been much slower and in itself probably more costly, I am convinced that had the faults and joint faces been carefully plotted in advance and drilling for blasting been so located as to avoid the major joints, it would have been possible to remove the proper amount of rock, so as to save the cost of rock removal from the river and leave a better and much safer face, at a lower final cost.

* State Geologist of Pennsylvania.

A. H. Coon,* Luzerne, Pa.—This highway project, 5½ miles in length from Easton to Martins Creek, along the Delaware River, was started in October 1929, and completed in August 1930, with the exception of the removal from the river channel of the excess rock which slid into the river after the tunnel shot at Paxinosa Point. This removal had been completed by August, 1931.

The occasion for the preparation of the data and exhibits indicating the structure of the rock cliff at Paxinosa Point, was a claim instituted by our firm, B. G. Coon Construction Co., of Kingston, Pa., against the Commonwealth of Pennsylvania for payment for the removal from the river channel of the excess material deposited therein by the slide which occurred after the "coyote hole" blast. The removal of this material had been ordered through the State Highway Department by the State Water and Power Resources Board, and authorization for payment therefor had been withheld by the Highway Department.

The claim was instituted in September, 1931, following completion of the work. After the first hearing before the Board of Arbitration in December, 1932, and following the introduction into the proceedings of a report on the geologic structure of the cliff by the State Geologist, our firm engaged Prof. W. B. Plank, of Lafayette College, to make a study of the structure, the methods employed by our forces to accomplish the desired result, and the reasons for the actual results. Professor Plank prepared a brief along these lines and argued the case in June 1933, before the Board of Arbitration, to a favorable decision rendered in November 1933.

I mention these facts only to establish the background. The "coyote hole" blast was fired Dec. 7, 1929.

B. F. Tillson,† Upper Montclair, N. J.—How was the rock removed from the river bed?

A. H. Coon.—By shovel. A dragline was first considered, but the material was eventually loaded by a 1½-cu. yd. power shovel and hauled out by trucks.

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* B. J. Coon Construction Co.

† Consulting Engineer.

Tunneling through Coal Measures with the Use of a Scraper Loader

By Gerald Sherman,* Member A.I.M.E.

(New York Meeting, February, 1936)

In the southern field of the anthracite region of Pennsylvania, many of the coal seams dip at angles of 40° to 50° from the horizontal. For transportation purposes, "tunnels" are driven across the coal measures and, occasionally, "rock gangways," beside the seams and parallel with them. The use of gangways driven in country rock appears to be increasing, as they avoid the maintenance of long gangways in coal, which eventually are driven to the colliery boundaries, and permit mining from a number of intermediate blocks without interference.

The rock consists of slates or sandstones of moderate hardness, which may range into very hard, highly siliceous, fine-grained conglomerates. In this distinctly bedded formation, the direction of the heading, whether parallel with or across the structure, makes it necessary to distribute the drill holes in a different pattern to break the ground, but does not change loading conditions.

In 1932, practically all the mucking was done by hand, and it was the established custom to use four men on two drills for drilling and breaking the ground, and five for loading it out. A driver and mule was needed in addition, to assemble the rock cars at a siding or branching track in the rear, pending their final disposal. This might occupy the driver's whole time, or if the length of haul were short, he might be able to do other useful work.

It was thought possible in 1932 to reduce the cost of driving such headings in rock by mechanical loading, and to investigate this subject a scraper loader was used in a section of tunnel then being driven. Operations while driving 195 ft. of this tunnel using a scraper loader were carefully observed. Time studies were made of the loading, and records made of the routine of breaking ground, but not in such detail.

Tunnels or gangways in the colliery are 7 ft. high above the rails, by 10 ft. wide, which requires a minimum of 8 ft. in height, broken to allow for rails and ties. This cross section is of sufficient size to permit the use of any one of several types of power loaders, but for this demonstration it was found most convenient to use a scraper loader.

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* Consulting Engineer, New York, N. Y.

The equipment consisted of three Ingersoll-Rand mounted drills, one jack hammer, drill steel, hoses, tools, etc.; a car for drills and tools, one water car, one Osana type loader driven by a 15-hp. a.c. motor, 440 volts,

FIG. 1.—MACHINE IN LOADING POSITION.

and one mule, as well as flexible 440-volt cable and coal cars for handling rock, 108 cu. ft. in capacity, standing 5 ft. 6 in. above the rails.

Fig. 1 illustrates the machine in loading position, Fig. 2 shows the car and the scraper slide extending over it, as it appears from the rear.

FIG. 2.—CAR AND SCRAPER SLIDE EXTENDING OVER IT, AS IT APPEARS FROM REAR.

The rock was a rather soft slate, easily drilled and broken, and the tunnel was driven squarely against the pitch. The bedding before blasting was not pronounced, but left somewhat saw-toothed longitudinal sections of roof and floor.

The loader was an old machine, rented for the occasion. It was not in particularly good condition, but caused no serious delays. The horizontal slide was too low to go over the cars. It was necessary therefore to jack up the rear end of the loader at each set-up, support the rear wheels on iron shims, and use special anchor bolts to hold the frame on the rails. This tilted the rear end of the slide upward and made it necessary to break the tunnel a little higher than the standard section. That would not have been necessary if a loader of recent design, suited to the tunnel section and cars, had been available. A new machine would have saved time in setting up, and the rate of loading would have been a little faster. Repairs were probably heavier than the average, but accurate data on repairs could not be assembled in driving 195 ft. It is concluded that with suitable equipment, the loading would have been done a little more quickly, but no more than enough to provide a small safety factor in a comparison with hand loading.

It was the object of the investigation to find out whether the cost of driving a tunnel by power loading would be cheaper than hand loading or would increase the rate of advance.

The cars of rock were hauled by a mule to switches at an average distance of 750 ft. from the face.

Operating Routine

The work was carried on in time cycles of two shifts per day. The drillers went on shift at 3:30 p.m. and the muckers at 7:00 a.m. on the following morning. Drilling and blasting could overrun the 8-hr. shift without delaying the cycle, but more than half an hour of overtime in loading would hold up the drillers.

Breaking Ground

The drilling and blasting crew consisted of one chargeman, two drillers and one helper. When the drillers went on shift they found tool and water cars at the end of the track, about 65 ft. back from the face, and the face reasonably well cleaned for setting up the drills.

Operating Cycle.—The operating cycle was as follows:

1. Drills were set up.
2. Round was drilled.
3. Drills were torn down.
4. Water and tool cars were pushed back out of danger from blasting.
5. The cut holes were shot.
6. Ventilation fan was started and compressed air turned on to blow out smoke and gases.
7. The break of the cut holes was examined. They were recharged when necessary and fired with the rest of the round, using instant and delay-action detonators.

8. The ventilation fan was left running when the crew went off shift.

The holes were drilled in a standardized pattern with six cut holes taking out a roughly pyramidal block from the bottom center of the face. Consecutive advance was:

Rounds	Feet per Round	Feet
10	6.7	67
6	7.5	45
5	7.4	37
6	7.67	46
—	——	——
27	7.22	195

For the last 128 ft., when longer drills were provided, 7.53 ft. were broken per round.

Advance per round was 7.22 ft. There were 18 holes per round, the average length of hole being 9.3 ft. Powder used per round was 97.6 lb. and per yard of advance, 40.5 lb.

The average time required was 2 hr. for setting up drills, 2½ hr. for drilling and 2½ hr. for tearing down, charging and firing. The least time for a completed round was 5¼ hr. but in that round no recharging of cut holes was necessary.

Some cut holes were recharged in 6 rounds, all cut holes were recharged in 6 rounds and none were recharged in 15 rounds (a total of 27 rounds). There were misfires in four holes.

The air pressure was rather low and the drills of an old type, but actual drilling occupied only about 2½ hr., or 30 per cent of the 8-hr. shift, and did not in any case interfere with completing the round within that time. The usual methods of drilling and blasting were followed, to avoid any conflicting influence on the comparison between hand and power loading.

The rounds were finished well within the 8-hr. shift; in one instance, as noted above, in 5¼ hr. By using higher air pressure and such drills as are now available, time would have been saved and possibly drill maintenance, but labor costs would not have been reduced, as the drilling crew went off shift when the round was broken.

By trying out different positions for the holes drilled, particularly of the cut holes, and using extra holes for the cut, the length of the round broken would probably have been increased, or the consumption of powder decreased. When the beds pitch toward the driller it would be better to take the cut from the top center of the face.

For any ground, there must be some pattern of holes and some length of round broken that are most efficient. If drilling is done on one shift and mucking on another, the most efficient round is the longest that can be broken without the excessive consumption of explosives. A longer round would more fully occupy both drilling and mucking shifts until too

much rock is broken to be loaded out in a shift. In fact, powder can be wastefully employed and reduce costs if enough extra footage is made. In this particular case, it would have been accomplished by drilling more and deeper holes and probably by using more powder per foot of advance.

In hard ground the advance per round broken is limited, and the consumption of powder per foot, and the number of drill holes to contain the charge, increase rapidly with any length of round beyond a natural break.

If speed is not a particular object, a three-man drill crew would be more efficient.

Mucking

The crew consisted of one machine operator and one helper, who also acted as driver.

The face was mucked out and track laid when necessary, every second or third day.

Operating Cycle.—The operating cycle in mucking was as follows:

1. Haul water and tool cars out to a siding.
2. Dress down roof and face.
3. Clear the track of fly rock.
4. Bring in loader.
5. Drill holes in face for eyebolts and key them in.
6. Set up machine.
7. Load.
8. Move loader back.
9. Clean up tunnel and ditches.
10. Lay track when necessary.
11. Bring in tool and water cars.

The operating time is summarized in Table 1.

The track must be cleared of fly dirt before the loader can be brought in. If the loader is set up less than about 65 ft. from the face, there is too much hand loading for two men. Hand loading in this tunnel would have required five men in the loading crew and at least one-half of a driver's time.

By using a scraper loader, the rock was loaded out and transported to the siding by two men. The expense incurred for power, and the maintenance and repairs of the loader and for interest and depreciation of the loader and flexible power cable must be added to obtain a comparison of costs (Table 2).

Loader repairs for the period of test, including rope replacement, amounted to \$82.63, or \$1.27 per yard. Electric power used per yard amounted to 9.106 kw-hr., charged at 10¢, or 24.1¢ per day.

The costs of getting the machine into the colliery and of laying the cable are not included because they were governed by local conditions. The cost of interest and depreciation is estimated from meager data.

TABLE 1.—*Operating Time Summarized*

	Average Day, Min.	A Day when Track Was Laid, Min.	A Day when No Track Was Laid, Min.
Preparing to load	88.4	84	97
Loading	225.7	230	226
Laying track		64	
Cleaning up and moving out	106.2	61	95
Bringing in drill equipment	14.0	14	14
	434.3	453	432
Cars loaded	10.1	10.2	10.0
Time required	7 hr. 14 min.	7 hr. 33 min.	7 hr. 12 min.

Cars loaded	By hand	9	Average per day	10.1
	By machine	263	Average day in maximum week	11.5
		——	Maximum	14.0
		272		

TABLE 2.—*Comparison of Costs per Yard of Advance*

	Power Loading	Hand Loading
Labor	$ 5.63	$12.48
Driver		1.25
	$ 5.63	$13.73
Loader repairs	1.50	
Power	0.10	
	$ 7.23	$13.73
Interest and depreciation of:		
Loader	0.75	
Flexible electric cable	1.00	
Cost per yard of advance	$ 8.98	$13.73
Cost per day	$21.61	$33.03

The cost of the flexible electric cable is unusually high because it had to be of a type approved for a closed-lamp colliery. If a tunnel is to be driven from a point far distant from a power line, the cost of the cable might prohibit the use of an electrically driven loader for a short job. Compressed air, which is always at hand when drills are used, would drive the loader at a greater expense for power but would cut out the cost for cable entirely. This is a matter that must be decided by the comparative cost for each project.

Two men operating a scraper loader could do little more than handle the muck from an advance of 7.5 ft. per round in an 8-hr. shift, but there is a certain flexibility in the system. When unusual delays are encountered, the muckers need only drag the rock back and clean up the face to prepare for the drillers. The loading would be done on the next shift.

A greater margin to cover emergencies would be obtained by using a loader of recent design built to suit the colliery requirements. Further improvements may be expected by perfecting the operating routine through the development of skill by experience, and experiments in the details of methods or equipment.

The number of distinct operations in the mucking cycle lends itself to the use of one or two or more men without an important loss in efficiency. In cleaning track and most other operations, three men could work as efficiently as two and in laying track they would do better, all with a corresponding gain in time. When loading there would be some idle time but a third man would shorten the loading period a little. A fourth man would be perhaps 50 per cent efficient.

The work done by operator and helper is shown in a series of charts (Figs. 3 to 12). Although the records covered each mucking shift in equal detail, only representative conditions are exhibited in the charts.

These charts are projected to illustrate two-shift time cycles using a scraper loader operated by two men, and continuous three-shift work cycles using four men on each shift, who either drill or muck as required.

When breaking a long round, fly rock from the cut may be thrown back for a great distance. It is understood that by setting up the loader behind it and reversing the scraper haul, fly rock can be dragged forward beyond the end of the track and loaded as usual when the loader is in its final position. No attempt was made to do so in this case, for reasons that had nothing to do with its probability of success.

When the end of the track is so far from the face, loading will cost a little more for the additional scraper travel, but it was found that the time of loading was not appreciably increased because the operator dragged a carload forward close to the loader while the filled car went out. A track ending at such a distance from the face requires greater care in setting the center and grade lines to guide the drillers. A part of this section of tunnel was driven too high and made more rock to handle. In one place a few feet of the bottom had to be cut down to grade. A shorter distance between the end of the track and face would also be more convenient for the drillers when carrying drills and mountings to and from the face.

Conclusions

1. An appreciable saving in the cost of labor can be made by using a scraper loader operated by two men.

2. Greater speed can be obtained by using three men on the scraper without a significant increase in cost, but not unless such a change is made in the cycle as would usefully occupy the time saved.

TABLE 3.—*Average Time Spent by Scraper Loader*

Operation	Operator	Driver
Preparation for loading		
Shifting drilling equipment	3.8	7.3
Cleaning track	10.3	7.1
Moving scraper	8.3	10.6
Shifting cars	0.5	13.6
Dressing face	19.7	5.0
Eyebolts	29.9	9.0
Setting and jacking up	1.3	10.2
Scraper clamps	8.0	15.3
Miscellaneous delays	6.6	10.3
	88.4	88.4
Loading		
Loading	72.5	1.2
Shifting cars		108.4
Cleaning face and ribs by hand	19.2	3.0
Changing pulleys	14.1	1.9
Eyebolts	10.8	5.2
Miscellaneous delays	29.6	19.6
Lunch	10.1	9.0
	156.3	148.3
Operation dragging	70.8	78.9
	227.1	227.2
Driver cleaning track while driving		
Cleaning up		
Dismantling Equipment, eyebolts, scraper, etc.	17.0	3.8
Moving scraper	12.2	11.7
Shifting cars	1.7	17.3
Cleaning track	31.3	30.4
Laying track	30.5	29.4
Hand mucking	7.1	3.9
Scraper	2.9	0.4
Shifting drilling equipment	9.6	18.0
Miscellaneous delays	5.8	3.9
Lunch	2.0	2.0
	120.1	120.8

3. The organization for the lowest cost and moderate speed would consist of two crews of three men in a continuous cycle, each crew drilling or loading when required. Greater speed would be obtained by four men

TABLE 4.—*Operator's Time*

MINUTES

Days	Number of Cars	Loading		Dragging		Changing Pulleys	Replacing Eyebolts	Cleaning Ribs and Face by Hand	Lost Time	Lunch	Total	Per Car
		Total	Per Car	Total	Per Car							
Sept. 19	8	51	6.4	54	6.8	14	6	9	14		148	18.5
20	11	66	6.0	75	6.8	14	17		39		211	19.2
21	11	74	6.7	72	6.6	35	12	7	12		212	19.3
Total	30	191	6.4	201	6.7	63	35	16	65		571	19.0
Oct. 3	12	90	7.5	101	8.6	15	8	28	38[a]	12[e]	280	23.3
4	9	80	8.9	62	6.9	15	6	21	45[b]		229	25.4
5	10	76	7.6	68	6.8	16	13	19	16[c]		208	20.8
6	11	87	7.9	76	6.9	20	8	35		9	235	21.4
7	11	94	8.5	52	4.7	9	20	72	18		265	24.1
8	13	89	6.8	89	6.8	32	4	21	51[d]	10[e]	286	22.0
Total	66	516	7.8	448	6.5	107	59	196	168	9	1503	22.8

[a] Working on scraper and rope.
[b] Missed hole shot, 31 minutes.
[c] Breaking lump.
[d] Putting new pulling rope on scraper, 29 min., repairing pulley, 12 minutes.
[e] No time lost. Driver operated scraper.

TABLE 5.—*Driver's Time*

MINUTES

Days	Number of Cars	Shifting Cars		Loading	Cleaning Track	Changing Pulleys	Replacing Eyebolts	Cleaning Ribs and Face by hand	Lost Time	Lunch	Total	Per Car	Distance Ft.
		Total	Per Car										
Sept. 19	8	84	12.0		64						148	18.5	676
20	11	115	11.5		70	3	9		14		211	19.2	676
21	11	109	10.9		77		5	2	3	16	212	19.3	691
Total	30	308	11.4		211	3	14	2	17	16	571	19.0	
Oct. 3	12	120	10.9	10	86	13	5	7	33[a]	18[f]	274	22.8	752
4	9	89	11.1		94			3	43[b]		229	25.4	765
5	10	101	11.2		85		4	6	12[c]		208	20.8	765
6	11	117	11.7		99	10	3	3	3		235	21.4	783
7	11	111	11.1		100		4	23	27[d]		265	24.1	783
8	13	140	11.7		84		4		56[e]	12[f]	284	21.9	797
Total	66	678	11.3	10	548	23	20	42	174		1495	22.7	

[a] Working on scraper and rope.
[b] Shooting missed hole 31 min. Waiting for car loaded at chute 12 minutes.
[c] Breaking lump.
[d] Repairing rope 10. Waiting for car loaded at chute 17.
[e] Putting new rope on scraper, 39 min. Repairing pulley.
[f] No time lost.

on each shift without a serious increase in cost. The work cycle is equally applicable to one, two, or three shifts in 24 hours.

Acknowledgments

It is with pleasure that the author expresses thanks to the Philadelphia & Reading Coal & Iron Co. for permission to publish data obtained in driving a rock tunnel.

The records were charted by Denis Agar, who also assembled the records in convenient form and constructed the charts.

The driving of the tunnel was done by the Ashland Division of the Operating Department, and it was due to their very willing and valuable cooperation that it was carried through without interruption and in a systematic and efficient manner.

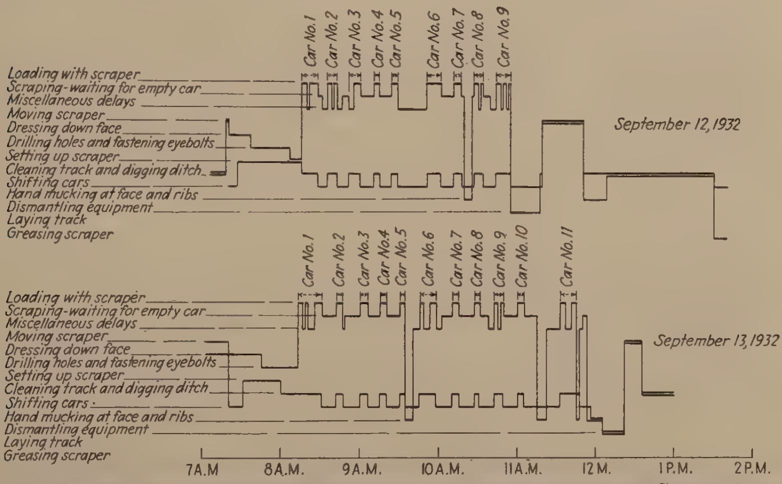

Fig. 3.—Charts of operations during entire loading cycles on Sept. 12 and 13. The two operators worked independently or together, as conditions arose.

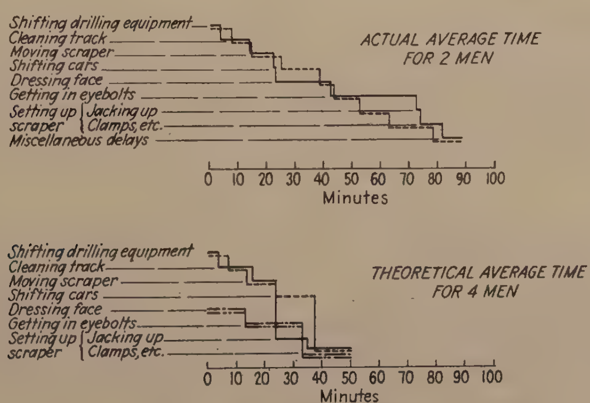

Fig. 4.—Chart of average work done in preparation for loading by two men, and projected chart to illustrate what might be expected from four men.

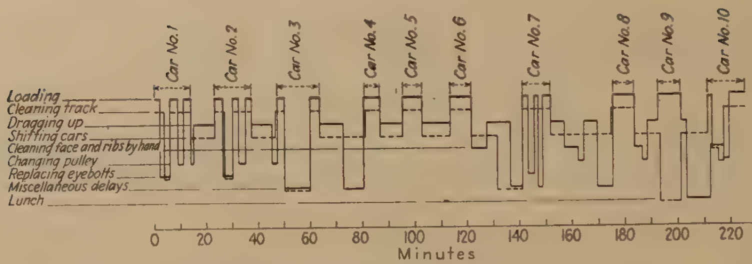

FIG. 5.—CHART OF AVERAGE LOADING PERIOD WITH TWO MEN.

Loading time would be shortened only a little by using more than two men. The extra men would assist in quicker shifting of cars and in some work at face, and in digging ditches and cleaning track when cars were available.

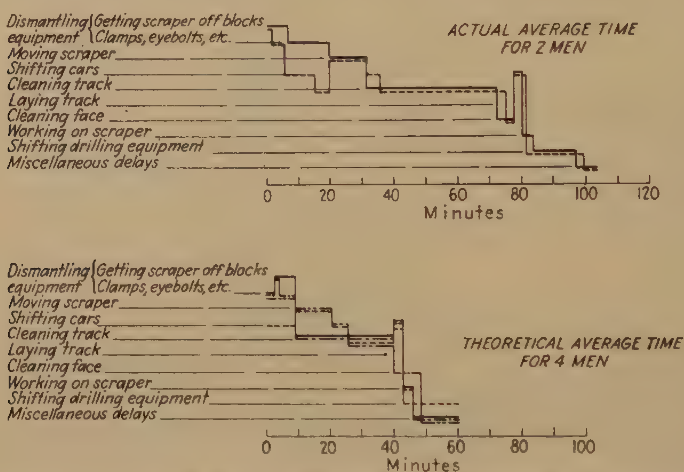

FIG. 6.—CHART OF AVERAGE WORK DONE AFTER LOADING WHEN NO TRACK WAS LAID AND THEORETICAL CHART TO SHOW WHAT MIGHT BE DONE BY FOUR MEN.

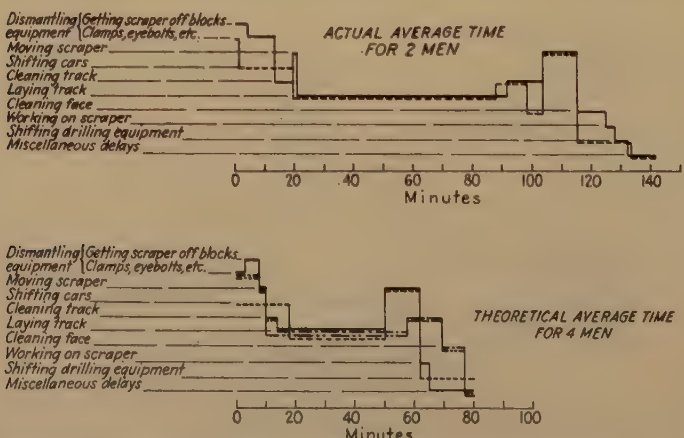

FIG. 7.—CHART AS OF FIG. 6 WHEN A SECTION OF TRACK WAS LAID.

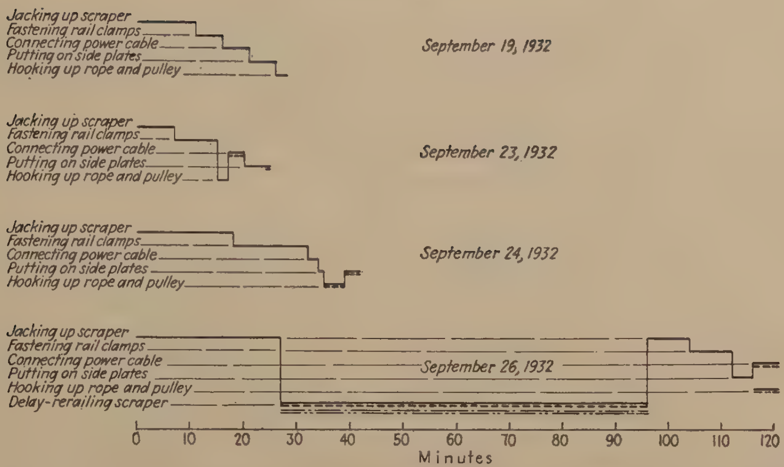

FIG. 8.—CHART OF DETAILS OF SETTING UP LOADER TO SHOW WHAT TIME MIGHT BE SAVED BY LOADER OF MODERN DESIGN. ADDITIONAL HELP WAS GIVEN WHEN LOADER WAS DERAILED.

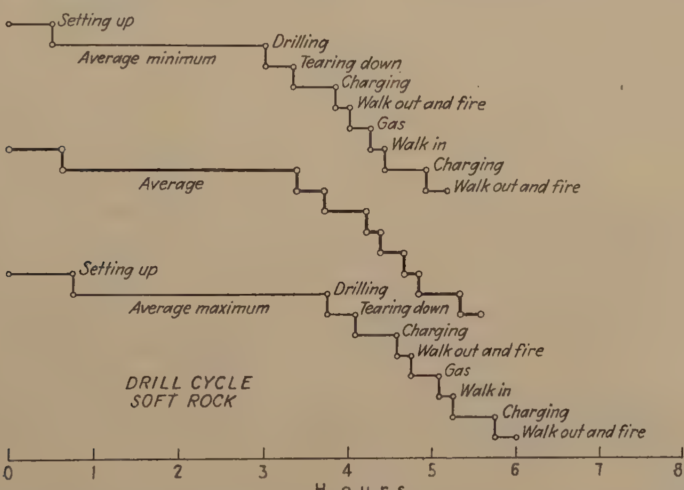

FIG. 9.—PROJECTED CHART TO SHOW DRILLING CYCLES IN SOFT ROCK FROM GENERAL OBSERVATIONS NOT ALL COVERED IN OPERATIONS DESCRIBED.

DISCUSSION

(John A. Church, Jr., presiding)

P. B. BUCKY,* New York, N. Y.—Labor and time studies have been made in South Africa, whence I have just returned. I find Mr. Sherman has been carrying on that very type of work here, and it is quite gratifying. The training there is such that they take a native from the wilds and make a good miner of him in a short time. Therefore I want to destroy one myth, and that is the superiority of the white working man, and the length of time it takes him to become a miner, because those natives

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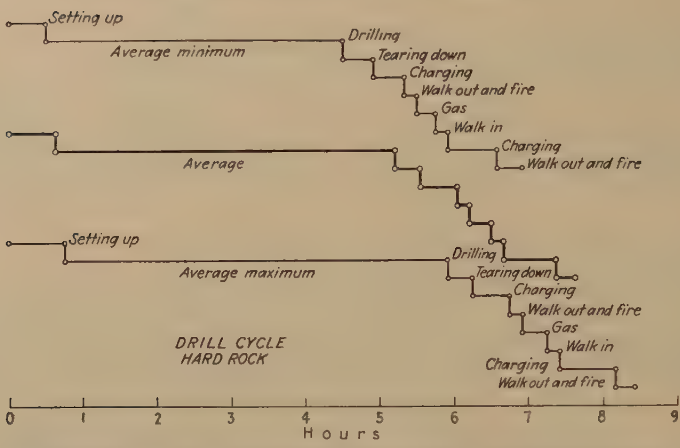

FIG. 10.—PROJECTED CHART TO SHOW DRILLING CYCLES IN HARD ROCK FROM GENERAL OBSERVATIONS NOT ALL COVERED IN OPERATIONS DESCRIBED.

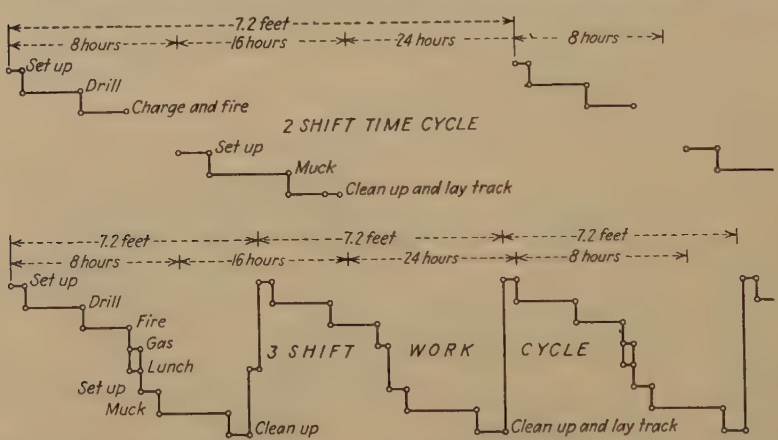

FIG. 11.—COMPARISONS OF TWO-SHIFT TIME CYCLES WITH SCRAPER LOADER AND TWO MEN, AND CONTINUOUS THREE-SHIFT WORK CYCLE USING FOUR MEN FOR SAME OPERATIONS IN SOFT ROCK.

become efficient quickly. Some of that efficiency may be due to the fact that when they go back to the tribe after their service period, they evidently tell the men that follow how to do things. It is quite common for natives to put in over 100 ft. of drill hole per shift on the Rand. These men can do it, and with labor at 87¢ a day there, one thinks twice before installing mechanical equipment.

G. SHERMAN.—I can confirm Mr. Bucky's remarks on the benefit of instruction for untrained men. During the war when there was a labor shortage, it was found that unskilled labor at times developed faster than partly trained men who had not learned fundamental principles.

J. L. G. WEYSSER,* Lansford, Pa.—Some difficulty was experienced in pulling the full length of the cut. Was any particular attention paid to the stemming used?

* Research Mining Engineer, Lehigh Navigation Coal Co.

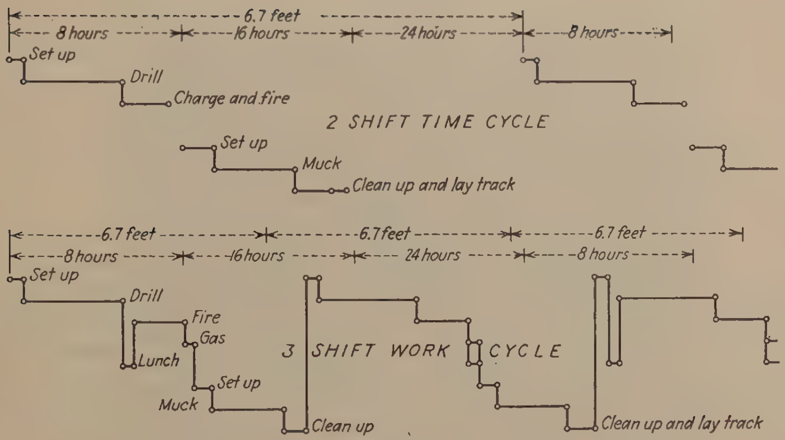

FIG. 12.—SAME COMPARISON AS FIG. 11, FOR HARD ROCK.

G. SHERMAN.—Yes.

J. L. G. WEYSSER.—I bring that out because in another portion of the Southern field where the work is done by rock contractors, no stemming was used unless they "used powder for stemming." (By "powder" I mean high explosives, not black powder.)

Another question concerns the scraper. I am not certain that I refer to it by the right term, but I believe the scraper used there is the box-type scraper. Were any other types of scrapers used?

G. SHERMAN.—Stemming was used. The failure to pull the full length of the holes drilled was probably caused by occasional errors in pointing the holes, or improper distribution of powder.

The scraper resembled the "hoe" rather than the "box" type. No trials were made of other scrapers or loading machines. It was desired to ascertain if any reasonably fast power-loading machine could be successfully fitted into the routine of rock tunneling.

J. L. G. WEYSSER.—I do not care to dispute the viewpoint on the use of stemming, but I might point out one experience I had recently: The Lehigh Navigation Coal Co. has been experimenting with rubber plugs for stemming, and in the process of introducing them it was found that in a great majority of the cases little or no stemming was used on rock contractors' jobs. By the use of some stemming, particularly with the rubber plugs, an appreciable increase in the advance and in the length of the cuts pulled was effected.

Power Loading on the Colorado River Aqueduct

By Arthur C. Green,* Member A.I.M.E.

(New York Meeting, February, 1936)

A group of 13 cities situated in Los Angeles and Orange counties in Southern California is engaged in constructing an aqueduct to carry water from the Colorado River at a point near Parker, Arizona, to a distribution point near Riverside, California. The aqueduct will have a total length of approximately 241 miles, and is designed to deliver 1605 cu. ft. of water per second, or a little more than one billion gallons per day. The aqueduct crosses a barren, desert waste and the project includes approximately 91 miles of 16-ft. tunnel.

The large amount of tunnel driving that has been done in this country in the past few years has resulted in some remarkable developments in equipment and methods. The building of the Colorado River Aqueduct, with its 91 miles of tunnels, has called on the best engineering knowledge in the country. New and better equipment has been designed for all phases of the work, and this equipment has been coordinated and adapted to the various conditions with a high degree of efficiency. The result has been that records for tunnel driving have been set under adverse natural conditions that exceed the estimates for progress under good conditions.

All of the tunnels in the main aqueduct, except the Valverde and Bernasconi at the western end, have a finished diameter, inside the concrete lining, of 16 ft. Fig. 4 shows a section of the tunnel with the concrete lining in place. Fig. 5 illustrates the finished tunnel in three typical sections. The standard tunnel grade is 3.432 ft. per mile in the 16-ft. tunnels, but the Valverde and Bernasconi tunnels have a steeper grade and a finished diameter of 15 ft. 3 in. The rough bore of the large main aqueduct tunnels averages 12.95 solid cubic yards of material per foot of advance in timbered sections and 10.76 solid cubic yards per foot in unsupported sections.

Tunnel Driving

While the Conway mucker has been used on tunnel work for years, a larger and more powerful Conway was designed to meet the require-

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* Goodman Manufacturing Co., Chicago, Ill.

FIG. 1.—CONWAY MUCKER AS SEEN FROM THE MUCK PILE.

FIG. 2.—CONWAY MUCKER DISCHARGING LOAD FROM DIPPER.

ments on this particular project. Figs. 1 and 2 show Conway muckers from the front end, and Fig. 3 is a view from the rear, showing the machine delivering muck to a car. The adaptability of this machine to the work is indicated by the fact that all but three of the 54 headings of the aqueduct have been driven with Conways.

Six-foot rounds taken from full-section tunnels and containing from 110 to 120 cu. yd. of loose muck have been loaded out in 40 min. In one heading two 10-ft. rounds were drilled and shot and three rounds were

FIG. 3.—CONWAY DISCHARGING MUCK INTO CAR.

loaded out in an 8-hr. shift. The time required for the mucking out of a round depends on a great many natural conditions and so varies widely on the various headings. A conservative average mucking time over all the headings on the aqueduct would be about 2½ hr. for rounds of from 9 to 10 feet.

The record for tunnel driving in rock is 302 ft. in one week of 21 shifts, or an average of 14.4 ft. per shift. The heading in which this record was established was advanced a maximum of 55 ft. in 24 hr. These records were made in a tunnel in which the rock was hard enough not to require support. A heading in which the material was slightly cemented gravel was advanced 315 ft. in one week of 21 shifts, or an average of 15 ft. per shift. The maximum advance in a rock tunnel for a 30-day period of

90 shifts was 1269 ft. This is an average of 14.1 ft. per 8-hr. shift. The 16,450 cu. yd. of material excavated in this 30-day period was all loaded by one Conway mucking machine. The tunnel in which this record was made was 100 per cent supported by steel ribs with wood lagging.

The average advance per heading over the entire aqueduct has been 18 ft. per day of three 8-hr. shifts. The dry headings have been advanced

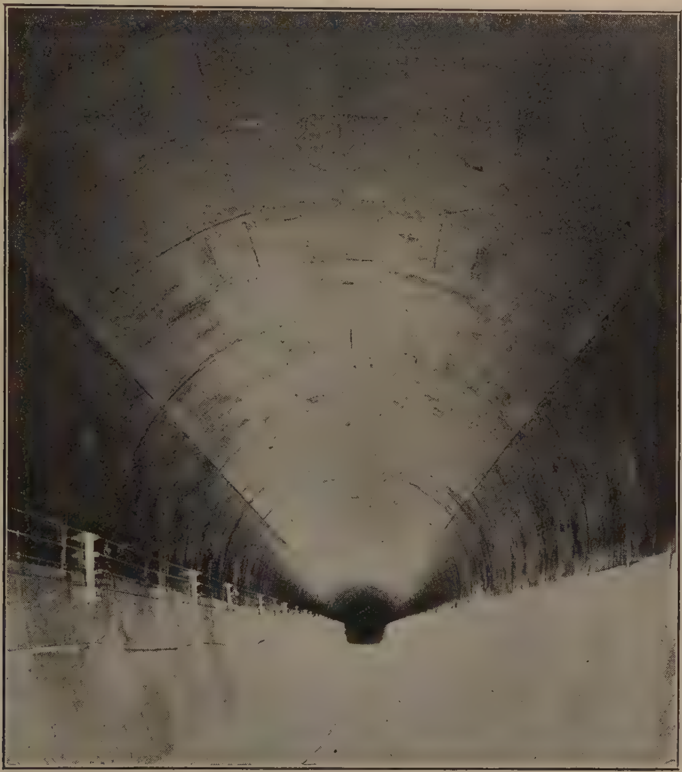


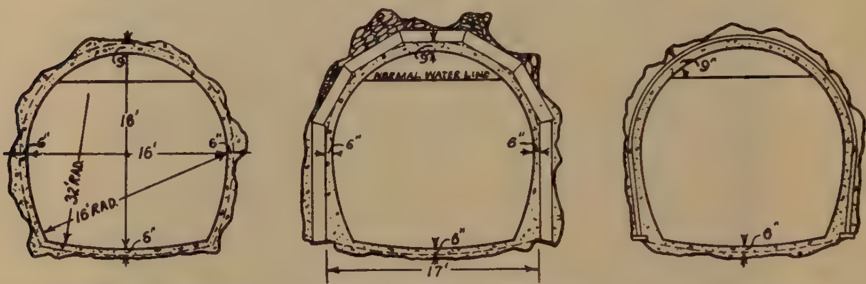
FIG. 4.—FINISHED SECTION OF TUNNEL.

at an average rate of 6.9 ft. per 8-hr. shift, and the wet headings have averaged 3.1 ft. per 8-hr. shift.

There has been a wide variation in conditions in the tunnels throughout the 241-mile length of the aqueduct. Material has varied from hard, blocky granite to fine sand, and some headings have been so wet that large pumping plants have been installed to handle the water, while others have been so dry that sprinkling has been necessary to settle the dust.

A high percentage of the tunnel driven has been supported. The supports used have been of three kinds: (1) Timber posts and five-segment arches with wood lagging; (2) steel ribs with wood lagging; (3) steel ribs with steel liner plates.

The type of support used has been governed in some instances by the character of the ground, and in some by the cost. The size and spacing of the supports has varied according to the ground. The size of timbers used has varied from 10 by 10 in. to 16 by 16 in., the average being 12 by 12-in. timbers placed on 6-ft. centers. A considerable amount of heavy ground has been encountered in both wet and dry headings, and in some cases the weight was so great that re-timbering was necessary before the concrete lining could be placed. In one instance



a. Untimbered section. *b.* Timber-supported section. *c.* Steel-lined section.
FIG. 5.—TYPICAL TUNNEL SECTIONS.

it was necessary to place 16 by 16-in. timbers as close together as possible to bear the weight.

Because of the dry nature of the country, the original estimate on the aqueduct included less than 10 per cent supported ground. It speaks well for the organization and equipment on the various headings that, although the actual amount of support has been about six times the original estimate, yet driving progress is generally ahead of schedule and costs are below expectation.

Costs

The ground through which the aqueduct is being driven is extremely variable, therefore the costs on the various jobs differ widely. A figure reflecting an approximate average cost per cubic yard of excavation in the dry tunnels, which are on force account, is $3.90. This average total excavation cost is made up of the following classifications:

1. Breaking (includes drilling, drill steel, sharpening, power, water, powder, and caps)	$1.25
2. Mucking (includes labor, supplies and power)	0.48
3. Hauling (includes labor, supplies, and power)	0.30
4. Maintenance (covers cost for the entire plant and equipment)	0.42
5. Plant (figuring 100 per cent depreciation for the job)	1.25
6. Camp maintenance (covers compensation, insurance and superintendence)	0.20

The lowest excavation cost on the dry force account tunnels was $3.30 per cubic yard, and the highest was $4.43 per cubic yard.

Some representative figures for total labor costs, for excavation and all incidental related activities are given in Table 1. These figures include the labor costs for all the work of drilling, shooting, mucking, hauling, timbering and maintenance. The average total labor cost for force account tunnels taken for 30 operations, including both wet and dry conditions and averaging 55 per cent supported, was $24 per lineal foot, or about $2 per cubic yard.

TABLE 1.—*Representative Labor Costs*

Condition in Tunnel	Percentage of Tunnel Supported	Total Labor Cost per Lineal Ft.	Total Labor Cost per Cubic Yard
Dry	65	$15.20	$1.38
Dry	44	25.50	2.32
Dry	100	17.20	1.38
Dry	100	20.70	1.65
Wet	26	22.30	2.03
Wet	12	50.60	4.60

Ninety-six men are considered a standard camp crew for one heading on the force account tunnels. These men provide three shifts of 8 hr. each for continuous operation and include all labor, of whatever classification, connected with the driving of that heading. The crew at work in a heading, of course, varies with conditions, but averages 18 to 20 men. The required minimum wage-scale classification on the aqueduct is as shown in Table 2.

TABLE 2.—*Wage-scale Classification*

Class	Wage per 8-hour Day	Class	Wage per 8-hour Day
Blacksmith	$6.00	Miner, machine man	$5.60
Compressorman	5.00	Miner, chuck-tender	5.20
Concrete finisher	5.60	Motorman	5.20
Concrete equipment operator	5.20	Mucking machine operator	8.00
Driller	5.00	Powderman	6.00
Laborer	3.80	Timber framer	5.60
Mechanic, repairman	6.00	Miscellaneous, not less than	3.20

Maintenance costs on the Conways on the aqueduct have varied greatly because of the great variety of conditions that have been met. In sections where conditions have been good and the machines have been regularly inspected and maintained, these costs have been as low as 4¢ per cu. yd., including belt replacements and all repairs. In other sections, where wet, coarse muck has been encountered or where the machines have not received proper care, the costs have run as high as 12¢ to 14¢ per cu. yd. The average maintenance cost over all the Conways on the aqueduct, including belt replacements and all other repairs, would be about 9¢ per cu. yd. The length of belt life on the

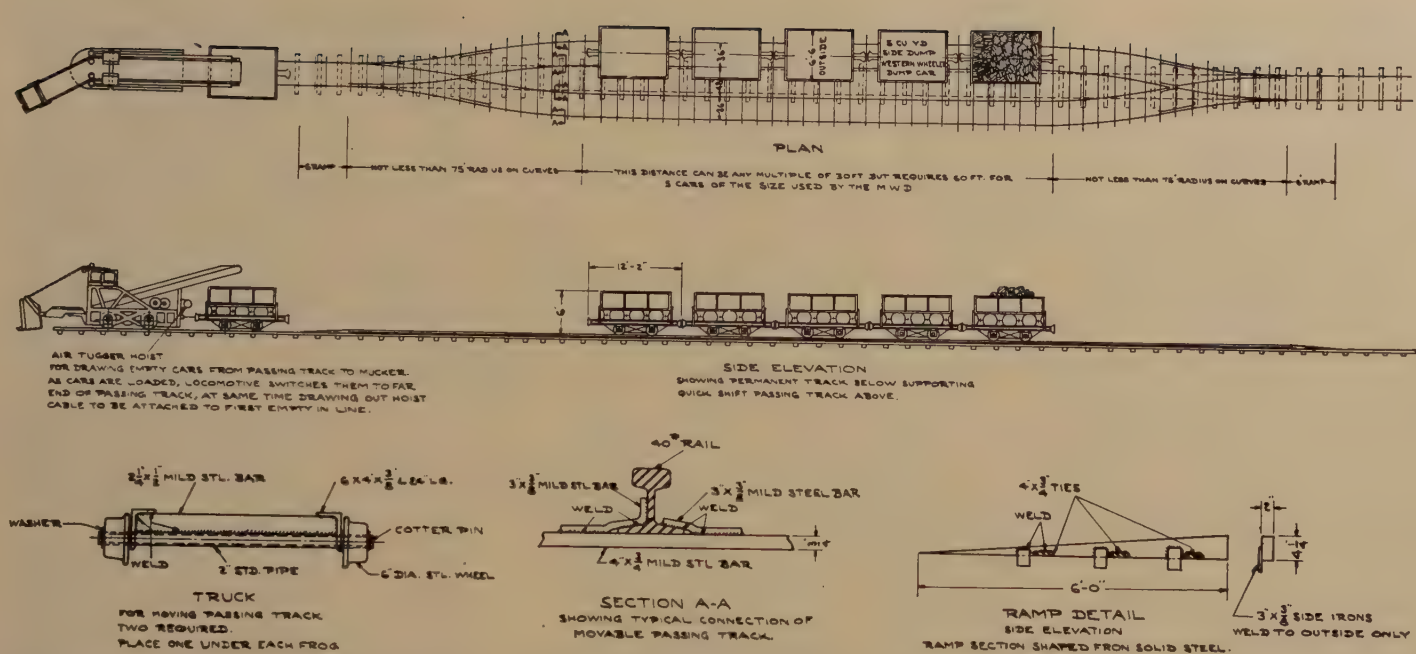


FIG. 6.—CALIFORNIA SWITCH.

Passing track shown here is easily and quickly moved by placing truck under each frog and pulling ahead with either mucker or locomotive. More than two trucks are not advisable as irregularities of permanent track cause trouble.

Permanent track must be exactly on center line of tunnel to allow car clearance with walls. Curves must be not less than 75-ft. radius to clear overhang of drill carriage. Passing track is supported through center by permanent track but requires blocking beneath outer rails.

Joints of all four rails should break evenly and be joined with standard splice plates. Rails are 40 lb. per yard. Ties are 4 by ¾-in. mild steel bar fastened to rails, as shown in sec. *A-A*.

Position of permanent track shown by dotted lines.

machine is dependent almost directly on the type of muck being handled. Belts handle from 10,000 to 30,000 cu. yd. of broken material before being replaced.

Car Change

The four principal systems of car change used on the aqueduct are: The California switch, the Cherry Picker, the Grasshopper, and the Dixon conveyor.

The California switch (Fig. 6) consists of a portable combination of siding and switch superimposed on the main track. It is provided with tapered end rails and a spring switch at each end of the siding so that the cars may pass directly from the permanent track up the tapered end rails to either side of the portable siding. The empty cars are pulled from one track of the siding up to the mucking machine by a small tugger hoist mounted on the rear end of the mucking machine. The loaded cars are pulled away from the mucking machine and on to the other track of the siding by a locomotive. There is very little work involved in moving this portable siding; so that it is possible to keep it very close to the heading, thus providing a very quick car change. It is moved by placing a low truck under the frogs at each end and moving the whole thing forward as a unit on the permanent track.

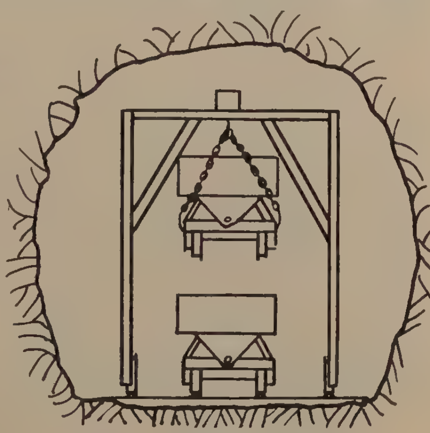


Fig. 7.—Cherry picker.

The Cherry Picker (Fig. 7) is a movable framework supporting an air hoist, which picks up an empty car from the main track so that the way is clear for a locomotive to pass and pull a loaded car away from the mucker. There are a great many modifications of the Cherry Picker system of car change. The empty car is sometimes elevated directly above the track and held there until the locomotive has passed underneath and pulled out the loaded car, and it is sometimes elevated from the track and placed at one side until the load has been pulled away from the mucker by the locomotive. The locomotive pulls a trip of empty cars up to the Cherry Picker and stops with the first empty car behind the locomotive directly under the hoist. This empty car is swung in the clear. The locomotive backs up to the other end of the car, the car is swung back on the track, and the locomotive pushes it up to the mucker. The locomotive then pulls the second empty car up under the hoist, where it is swung in the clear. When the first car is loaded, the loco-

motive pulls it out beyond the Cherry Picker, the second empty is placed back on the track ahead of the load, and the locomotive pushes the empty car together with the loaded one up to the mucker. This process is continued until the entire trip of cars is loaded, each empty being elevated by the Cherry Picker and transferred to the front end of the trip. Five headings on the aqueduct have the Cherry Picker car passer with drills mounted on the front end of the framework, so that it is made to serve also as a drill carriage.

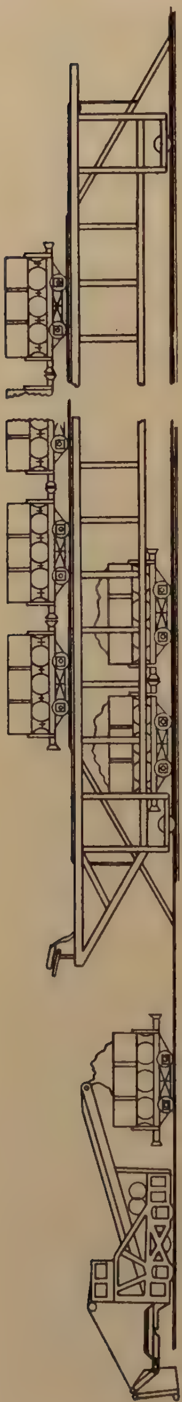


FIG. 8.—KING GRASSHOPPER.

The Grasshopper, or King car passer (Fig. 8), consists of a steel frame about 150 ft. long, traveling on rails set on each side of the main-line track and on 10-ft. gage. There is a hinged ramp at each end of the framework, which is operated by a small air hoist, and track is laid up these ramps and over the top of the deck of the framework. Fig. 9 shows the back end of one of these Grasshoppers with the ramp down in position for pulling the empty cars up on top of the deck. Fig. 10 shows this same end with the ramp elevated so that the locomotive can pass out from under the deck with the loaded cars. The car change is accomplished as follows: (1) six to eight empty cars are pulled up the rear ramp by an air hoist and stored on the deck; (2) they are lowered one at a time down the front ramp to the rear of the mucking machine; (3) the front ramp is then raised and when the car is loaded, it is pulled back under the deck of the Grasshopper by a locomotive; (4) the front ramp is again lowered and the process is repeated. The Grasshopper has a cantilever beam extending out over the front ramp, on which the drills are mounted. The deck of the beam also makes a convenient working platform for the timbering crews. The device, therefore, serves three purposes: it is used as a car passer, drill carriage and timber jumbo. Seven headings on the aqueduct have been driven using this Grasshopper as a car passer, drill carriage and timber jumbo.

The Dixon conveyor (Fig. 11), consists of a long conveyor belt mounted on a framework that travels on wide-gage rails in a manner very similar to that of the Grasshopper. The conveyor is long enough to

permit six to eight empty cars to be pushed underneath it. The front end of the conveyor operates on a hinged ramp, the lower end of which widens out into a hopper. This hopper is constructed long enough to give the mucker ample space in moving toward and away from the muck pile in cleaning up the heading. The conveyor carries the muck from this hopper and drops it into the cars as they are pulled out from under the conveyor belt at the rear of the framework. The front end of the framework extends out over the hopper in a cantilever beam on

FIG. 9.—KING GRASSHOPPER WITH RAMP DOWN.

which the drills are mounted. The conveyor belt is mounted high enough over the track to permit the mucking machine to pass underneath; so upon completion of the mucking cycle, the hopper is elevated, the mucking machine is backed away from the face and drilling is begun immediately. Only one heading on the aqueduct was driven its entire length using an auxiliary conveyor.

The choice of car-passing equipment in various headings of the aqueduct has depended mainly on the personal viewpoints and ideas of the engineers or superintendents on the job. The California switch is the simplest of the devices used and has been adopted as standard on the

district's Force Account tunnels. It has the widest range of adaptability and it has been used on the majority of the headings. The other types of car-passing equipment require a great deal more room and in variable ground sometimes slow down the drilling and timbering operations. Car change over the entire aqueduct varies from 20 seconds to about 2 minutes, but the average would be about 40 seconds.

FIG. 10.—KING GRASSHOPPER WITH RAMP UP.

In headings where the drill carriage is not incorporated with the car passer, the drills are mounted on a drill jumbo; that is, a framework about 20 ft. long mounted on a car chassis with a long wheel base and hinged decks that can be opened to provide working space for the drill crews during the drilling operation. When the drilling has been completed these decks are folded down on the central framework and the jumbo is pulled out to a siding by a locomotive and left there until the next drilling cycle. Fig. 12 shows a drill jumbo with the decks out and the drill crews in place. From four to six automatic pneumatic feed drills are mounted

on the front end of the jumbo and it is equipped with the necessary piping and hose connections for water and air for the drills. The jumbo is also equipped with permanent wiring and lights. Jumbos furnish not only a convenient method of transporting the drills and placing them in position for their operation, but they also are convenient carryalls. Tools and working material are carried on the jumbo, so that they are always within easy reach of the crew.

Drilling and Blasting

The number, depth and spacing of the holes varies with the nature of the ground and often varies from one round to the next. The drilling is determined by the men on the job, by experimentation with the different combinations until satisfactory shooting is obtained. Full-section headings require from 18 to 60 holes per round drilled to a depth of from 6 to 13 ft. The average number of holes required per round in a full-section heading is about 45.

Shooting has all been done electrically with a line voltage of 440 volts. Special shooting lines and interlocking switches have been installed and used according to the most stringent safety practice. The number of delays used and the time spacing between delays varies according to the ground, and the loading of the holes has been strictly controlled. On all force account work, and on many of the contract jobs, a special safety primer developed by the Metropolitan Water District has been used. Dynamite of 40 per cent strength was most commonly used, but in certain conditions some of 25 and 60 per cent strength was used. The quantity of explosives has varied from 1 lb. per cu. yd. in cemented gravel to 7 lb. per cu. yd. in hard, dead granite. The average quantity over the entire aqueduct would be about 3½ lb. per solid cubic yard.

Concreting

When the tunnel excavation has been completed, work is begun on the placing of the concrete lining. This lining is placed in separate sections, as shown in Fig. 13; the curbs *A* being first, the arch *B* second and the invert *C* last. Each of these sections of lining is

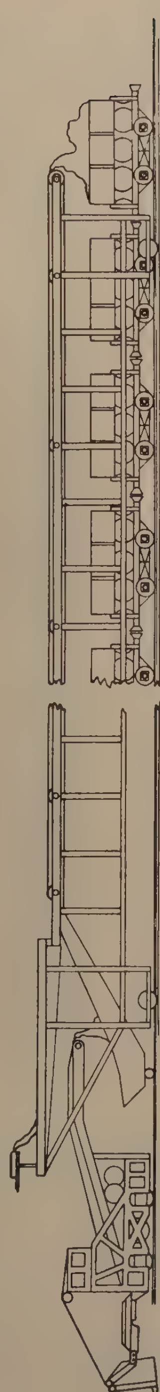


Fig. 11.—Dixon conveyor.

begun at the point farthest from the aggregate plant and is completed before the next section is placed.

The curbs are located and lined up accurately by a transit and later are used as supports for the steel arch forms, thus doing away with the need for further instrument work. The forms used in placing the arch lining are made up in 30-ft. lengths and are constructed so that they may easily be collapsed on a carriage to such dimensions that they may pass under other forms that are in place in the tunnel. Fig. 14 shows a

Fig. 12.—Drill carriage with crew in position.

section of arch forms collapsed on a carriage for moving. The carriage is equipped with hydraulic jacks, which are used to expand the forms into position, where they are braced and held in position between the curbs by screw jacks placed between the forms and the haulage track. Fig. 15 shows the details of this bracing. About eight of the 30-ft. sections of forms are required at each concrete-pouring operation. As the concrete progresses, these forms are removed from the sections of lining that have set and are collapsed and moved out ahead of the freshly concreted sections to be used again.

Several schemes are used in mixing and placing the concrete. The force account tunnels are all being concreted by the Rex Pumpcrete machine (Fig. 16). The batches are proportioned by weight in quantities of 1 cu. yd. each at aggregate and batching plants outside the tunnel, are placed in specially designed batch cars and pushed into the tunnel and up to the mixer by a locomotive. A hoist on the mixer elevates the body of the batch car and dumps its contents into a hopper on the machine, from which it drops into the cylinder of the mixer. The mix is dumped from the mixing cylinder into an ingenious pump,

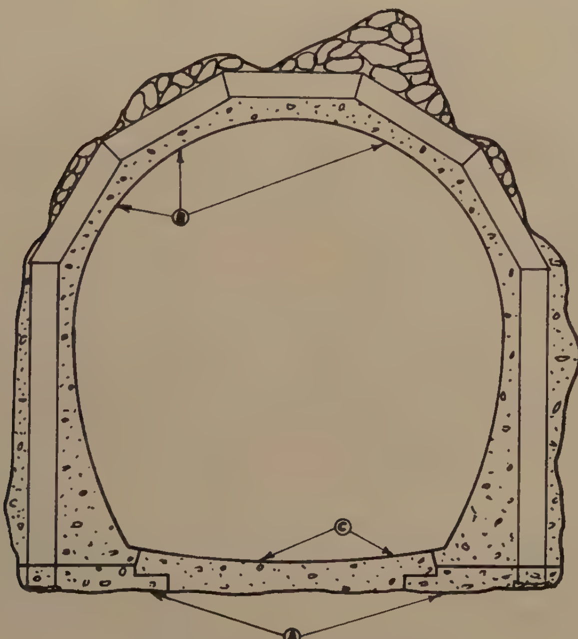


FIG. 13.—ORDER OF CONCRETING.

which forces the concrete through an 8-in. pipe line up over the tops of the forms to the advancing arch of fresh concrete. This pipe line is extended out from the boom shown on the front of the Rex Pumpcrete machine in Fig. 16.

The forms have hinged doors (18 by 24 in.) throughout their length so that inspectors can observe the concrete as it is forced into place and can ascertain that no voids are left. As the concrete arch advances, these doors are closed and the inspectors move ahead to the door next in line. These inspection doors are shown in forms in Fig. 14. The trips of batch cars are switched behind the mixing machine on a California switch.

Another method is to wet-mix the concrete outside the tunnel and transport it to the placing machine inside in batch buckets carried on

special cars, which are pushed by locomotives. These batch buckets are elevated from the cars by an air hoist mounted on a frame similar to that of the Cherry Picker, and their contents dumped into the cylinder of the placing gun. The cylinder opening is then closed and compressed air is introduced into the cylinder at a pressure of from 80 to 125 lb. per sq. in. This compressed air forces the concrete from the cylinder

Fig. 14.—Concreting forms collapsed on carriage.

out through the pipe, which is laid over the forms in the same manner as the pipe line of the Pumpcrete machine.

A modification of this system is used in several tunnels, whereby the dry mix is hauled to an inside mixing plant in batch buckets all on special cars pushed by locomotives. The batch buckets are dumped into the cylinder of the mixer, where water is added and thoroughly mixed with the batches. On some operations the equipment used is designed so that the concrete is forced directly from the cylinder of the mixer out through the pipe into the forms. Others use a placing gun, the concrete being dumped from the mixer into the receiving chamber of the gun.

The receiving chamber is sealed and the concrete is forced through the placing pipe out into the forms by the introduction of compressed air into the receiving chamber.

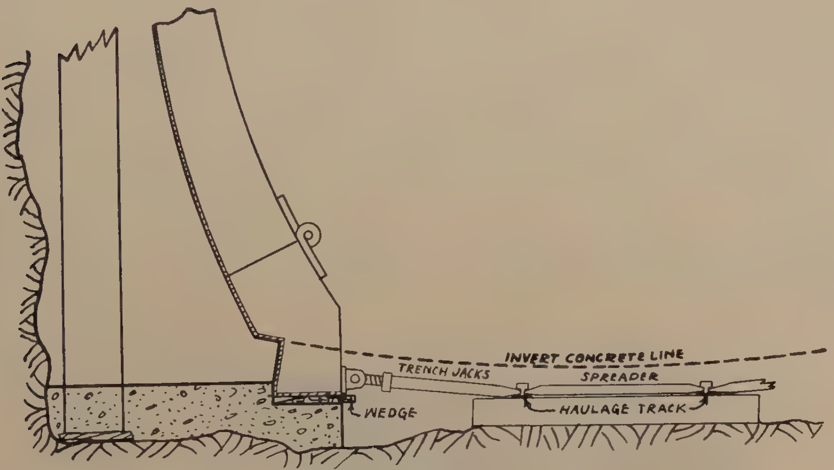


FIG. 15.—FORM DETAIL.

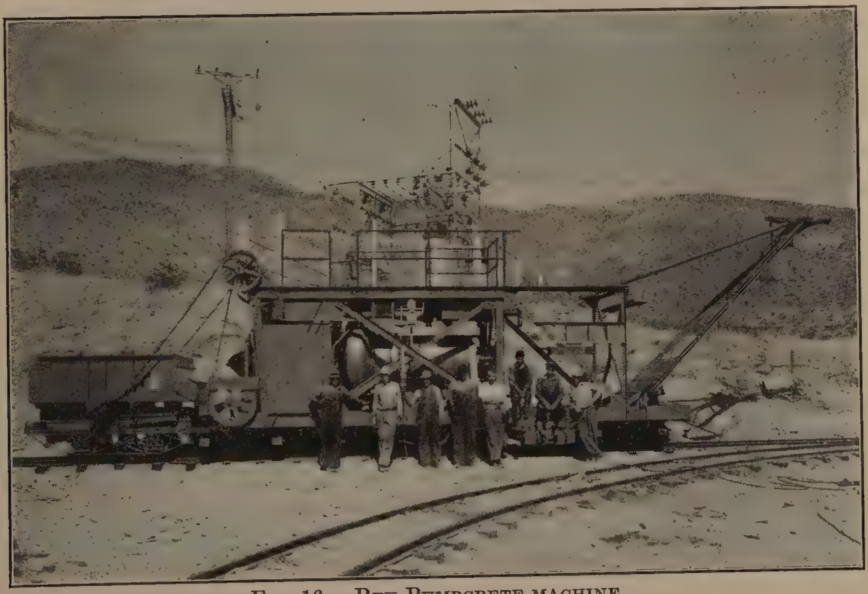


FIG. 16.—REX PUMPCRETE MACHINE.

As much as 735 cu. yd. of concrete has been placed in one tunnel in one day. In one month of 27 working days, 14,400 cu. yd. of concrete was placed; an average of 535 cu. yd. per day.

Ventilation

Ventilation in the tunnels of the aqueduct is provided by blowers of various sizes and of both the centrifugal and positive pressure types. The pipes used in connection with these blowers are of No. 14 gage electric-welded sheet iron and are 20 and 22 in. in diameter. Usually, fresh air is blown to the face, but for a short period after the shots have been fired the air is reversed and the fumes resulting from the shooting are sucked out. By reversing the air in this manner, the men may return to the face within 5 to 10 min. after the shots have been fired. Occasionally booster fans and short lengths of cotton tubing have been used at the face. Booster fans have also been used near portals and adits, blowing outward to accelerate the outgoing air. Some cotton tubing has been used with forced current for the full length of the tunnel, but

Fig. 17.—Combination trolley and battery locomotive.

inasmuch as the fumes must be driven out the full distance from the face to the portal, this method is not very satisfactory, except in short tunnels.

Haulage

Experience gained in planning haulage from the tunnels of the aqueduct and watching the manner in which these plans worked out has shown that it is very necessary to plan the haulage carefully and to take into consideration the most severe conditions that may be met. The whole operation in tunnel driving is dependent upon each part of the cycle being done in the shortest possible time. For efficient operation, at least two locomotives must be used per heading, where the headings are driven independently. The weight and type of these locomotives are determined by the maximum amount of muck per round, the mucking speed, the number of cars and the maximum length of haul. In straight battery haulage the size of the batteries is determined by the maximum amount of muck that can be excavated in an 8-hr. shift.

Most of the locomotives on the aqueduct have been of the storage-battery type. On the longer hauls, however, where more speed and greater capacity was desired, trolley and combination locomotives have been used. Several trolley and reel locomotives were used on the aqueduct. On the main line these locomotives receive their power through a trolley wheel, and in the part of the tunnel beyond the end of the trolley wire they receive their power through a cable that is connected to the end of the trolley wire. This cable is wound on a reel mounted on the end of the locomotive. The reel is power driven and pays the cable out as the locomotive goes in and winds it up as it returns. A number of combination trolley and battery locomotives were used, such as the one illustrated in Fig. 17. This type of locomotive has the higher speed and capacity that is characteristic of the trolley locomotive, when it is operating on the long main hauls where trolley wire is hung. It has the convenience of the storage-battery locomotive, when it is operating on sidings and in the part of the tunnel beyond the trolley wire, since it receives its power from the storage battery. The batteries receive charging current from the trolley wire when the locomotive is operating on the trolley, therefore they require an equalizing charge only about once a week.

A great many different types and sizes of cars have been used in the various headings. Both wooden and steel cars of the plain type and side-dump type have been used, ranging in capacity from 2 to 5 cu. yd. A few have had plain bearings, but the majority have had roller bearings. Experience has shown that the larger side-dump cars equipped with roller bearings are most efficient. The Metropolitan Water District purchased 5-cu. yd. steel side-dump cars with roller bearings for use in all the force account tunnels.

A Successful Dragline Dredge

By James F. Magee,* Member A.I.M.E.

(San Francisco Meeting, October, 1935; New York Meeting, February, 1936)

There is nothing new about dragline dredging for placer gold. The use of the separate unit for excavating preceded the large barge with excavator mounted upon it, which has reached a high state of perfection through use in every part of the world. But at present there is a vogue for a small, inexpensive plant combining a dragline excavator on caterpillar mounting and a gravel-washing plant on a floating barge. The most efficient ones use a bucket of 1 to 2 yd. capacity with boom from 40 to 70 ft. long. The barge varies from 20 by 30 ft. to 36 by 48, and carries the trommel screen, pump, riffle sluices and power engine.

Plants in Oroville District

In the Oroville district eight such plants have been built since 1933, and have been operated with varying success. The most successful of these is the plant of the Wyandotte Gold Dredging Co. This organization of six active partners constructed its first plant on Wyandotte Creek in the autumn of 1933. To September of 1935 about one million cubic yards of gravel was dredged for an output of about $225,000 in gold. The operation has been successful from the beginning, because of wise management, which was based upon good plant design and proper principles of operation. All the partners have taken a more or less active part in the operation. Usually there is one at work on each shift, and they have conducted their own prospecting and building also. They elected H. F. England as manager, with wide power of control, and he carries out this responsibility energetically and successfully.

One of the other plants, started shortly before the Wyandotte, finally abandoned its operation and sold its ground to the latter company, which successfully worked it for a profit. The failure of the other plant was due principally to poor excavating equipment.

Another organization that built a plant some months later than the Wyandotte company had considerable difficulty in management and was unable to keep ahead of expenses. Finally, late last summer, it entered into an agreement with the Wyandotte company to have the latter supply

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a dragline and dig its gravel on contract, at 7¢ per yard; also to undertake its management and remodel its dredge, for 25 per cent of the net income. This arrangement was quite successful, all outstanding obligations were cleared up and a real profit was realized for the first company. The Wyandotte company cleared the price of its shovel and a good profit for its management.

In the summer of 1934, Mr. England superintended the building of a dredge for an organization that operated for about eight months on ground not quite rich enough. The dredge was successful in its part of the operation. This ground was abandoned and the plant moved to the Redding district, where the operation is more successful as a whole, owing to richer gravel.

With this background of experience, the Wyandotte company built a new plant in the spring of 1935, on a new and larger piece of ground that was proved up for two years of continuous operation. This paper describes this new plant and its operation since it started on April 5, 1935.

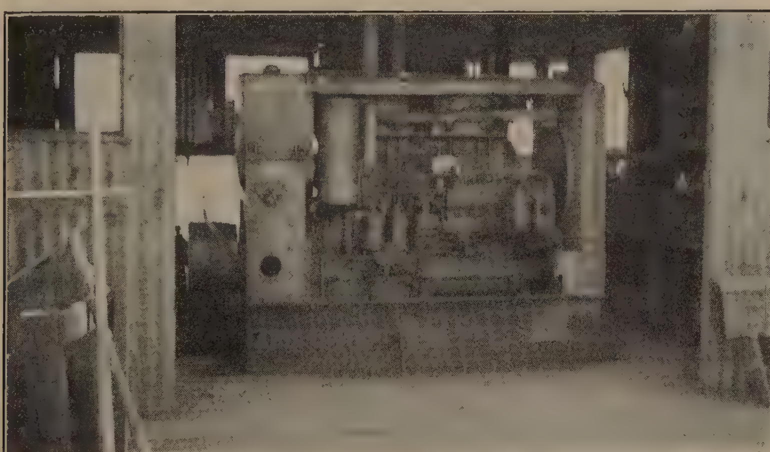

FIG. 1.—CATERPILLAR DIESEL POWER UNIT, UPPER DECK.

WYANDOTTE DRAGLINE DREDGE

Dragline

The original dragline was a Koehring 501. For this job its 45-ft. boom was lengthened to 50 ft. Its gas engine had been previously replaced by a caterpillar 75 Diesel. By changes in the gearing its speeds were increased. The original hoist speed of 162 ft. per minute was increased to 187 ft. per minute. The drag speed was increased from 125 ft. to 162 ft. per minute. The swing speed was increased from 3 r.p.m. to 5 r.p.m.

Buckets.—Three types of buckets have been used. Each has some distinct advantage. The first was a Page standard 1¼-yd. with four

teeth. Then a Page Heavy-Duty was used, which gave better life. The digging characteristics were not improved, however. Later, another Page standard was used. Then an Essco bucket with five teeth was tried. This bucket was narrower, longer and heavier, with greater pitch to the dipper front and teeth. It would dig harder ground but it would not fill as well as the Page and was more difficult to cast and to dump. For the present class of digging, the Page seems to be best adapted.

Engine.—The original gas engine, a Wisconsin, was exchanged for a caterpillar Diesel (Fig. 1) six-cylinder, 75-hp. power unit (D11,000). This engine has been in service 15 months and has been very satisfactory. It is economical, having cut the fuel bill to one-third the former cost for the gasoline engine, and is very dependable, having caused very little lost time in digging.

Lima Drag Line.—As there will be deeper ground to dig, making it advisable to have a longer boom and more power, the Koehring was sold recently and a new Lima dragline, type 601, was purchased. The boom is 60 ft. long. The motor is a caterpillar Diesel six-cylinder 102-hp. power unit (D13,000). The hoist speed is 180 ft. per minute; the drag speed, 170 ft. The swing speed is 4 r.p.m. For the contract digging, a P. & H. 600 with a 75-hp. Buda Diesel engine was used. It was satisfactory for that work.

Capacity.—In average digging two heaping buckets per minute are loaded continuously—more than 150 cu. yd. per hour.

Washing Plant

The washing plant, mounted on the hull, consists of a hopper, screen, riffle sluices, pump and stacker. The hull is of 3-in. pine planks over seven framed bents transverse to the length. There are two lines of crossed diagonal bracing and four longitudinal beams of 8 by 12, to take care of the working stresses from dumping into the hopper. The hull is 40 in. deep and when fully equipped and loaded draws 25 in. of water. The displacement is 65 tons.

Hopper.—The hopper is made of ½-in. plate welded together, with a flange on all four sides, for its suspension. The top opening of the hopper is 10 by 10½ ft. It is covered by a grizzly of seven inverted rails of 80-lb. section spaced on 16-in. centers. The flat bottom of the hopper has a slope of 25° toward the discharge chute, which is 30 in. wide.

Trommel Screen.—The trommel screen, built by the Link Belt Co., San Francisco, is 48 in. in diameter, 25 ft. long. The first 4 ft. and the last 5 ft. are blank. The middle 16 ft. are divided into four sections, each 4 ft. long, punched with ⅜-in. holes on 1⅛-in. centers; that is, ¾-in. bridge. The whole is made of ⅜-in. high-carbon plate and all joints are welded. The drive shaft drives both front and rear rollers on the left-hand side, which turn the screen by friction.

Screen Perforation.—There has been considerable experimenting with the screen-perforation pattern. The first pattern used was $\frac{3}{8}$ in. punching on $\frac{3}{4}$-in. centers; that is, $\frac{3}{8}$-in. bridge. The upper sections wore fastest and the first riffle sluice was most heavily loaded. Then the first upper perforated section was punched with $\frac{3}{8}$-in. perforation on $1\frac{1}{8}$-in. centers; that is, $\frac{3}{4}$-in. bridge. This distributed the gold better to the first, second and third riffle sluices, the load on the second being the heaviest. The wear was also much better distributed, because the wider bridge wore longer and the lower sections got less wear from sands. But the last two sluices, the sixth and seventh, still received a small proportion of sands. The pattern now used is $\frac{3}{8}$-in. punching on $1\frac{1}{8}$-in. centers throughout. The washing is satisfactory for the greatest part of the

FIG. 2.—DESIRED RIFFLE ACTION IN SLUICES. TROMMEL SCREEN AND WASH-WATER PIPES AT REAR.

gravel. (Fig. 2.) When, occasionally, a large proportion of silt is handled screen perforations become clogged to some extent. Usually this material is dug only along the margins of cuts already worked. Swampy places also sometimes contain excessive silt. As such material carries few values, these unusual conditions do not require any change to meet them. The wear on the screen is cut down considerably by the wider spacing of the perforations and a better distribution of the sands to the riffle sluices is obtained.

Screen Data.—The screen revolves clockwise, looking forward, at 14 r.p.m. The rate of travel of the gravel through the screen varies with the speed of feeding and the kind of material. It is difficult to time the travel of the ordinary gravel because there is no recognizable marker. A large rock often travels the 25 ft. in 30 seconds but sometimes it takes longer. Through the lower, smooth, blank portion of the screen the washed material travels at the rate of 50 ft. per minute uniformly for all

classes of material. Above the retarding rings, apparently, it travels at about ½ to ¼ of this speed. The screen has a fall of 1½ in. to the foot. In fast digging the screen may be half filled with material, which is over 10 cu. yd. of material, but such loading is unusual. Even under such conditions the stacker tailings are usually well washed.

Water.—The water is supplied by a Yuba centrifugal pump with 8-in. suction, 7-in. discharge, belt-driven at 1000 r.p.m. (Fig. 3). This pump delivers 1200 gal. per minute—equivalent to 6 cu. yd. per minute. The intake is through an 8-in. foot valve in a screen box of steel angle framing bolted to the side of the hull with a space of 1 ft. between it and the hull, so that four sides and the bottom are effective for screening the intake water. The box is covered with 3-mesh screen and there are

FIG. 3.—BELT-DRIVEN PUMP ON LOWER DECK OF WASHING PLANT.

75 sq. ft. of effective submerged area. A skimmer jet is used also to keep the surface trash away from the screen box most of the time.

The water is distributed about 84 per cent to the wash or spray pipe in the trommel screen, 14 per cent to two nozzles spraying into the hopper and 2 per cent to the skimmer jet. Forty miner's inches of water are bought from the irrigation company for use of the dredge.

Burden of Water.—An estimate of the burden of the water is based on an estimate of the sand ratio in the gravel, which varies, and the rate of digging, which also varies. In the tailings piles a fair approximation of volumes gives 40 per cent sand tailings to 60 per cent rock tailings. At a digging rate of 2½ cu. yd. per minute the sands would amount to 1 cu. yd. per minute. The water amounts to 6 cu. yd. per minute, giving 15 per cent as the average amount of sand in the pulp. Random tests of the pulp gave from 5 to 20 per cent sand, the first sluices carrying the heaviest sand burden.

Riffle Sluices.—The riffle cross sluices are 27 in. wide inside and there are seven cross sluices on each side of the screen. These discharge into a side sluice, 4 ft. wide, on each side. The side sluices terminate in steel flumes, which project 12 ft. beyond the rear end of the barge, carrying the sands well away. The sluices are lined with riffle sections of the Oroville type, wooden bars covered with a strip of strap iron slightly wider than the wooden bar. The sluices are all on a grade of 1½ in. to the foot. There are 537 sq. ft. of sluice area, of which 480 sq. ft. are net effective riffle area.

Stacker.—The coarse washed material is discharged from the trommel screen on to a 24-in. stacker belt (Link Belt Co.), which travels at a speed of 200 ft. per minute on a steel stacker bridge 45 ft. long. The bridge is pivoted on the foot pulley and suspended by a cable, over a gantry, tied on to the hopper.

Engine.—The power plant is a caterpillar Diesel, four-cylinder (D7700) power unit operating at 850 r.p.m. with a rating of 50 hp. for sustained load. The drive is by leather belt to a countershaft from which the pump, screen and stacker are driven by belt or chain drive. These engines were secured from the Butte Tractor Co., of Chico.

Other Equipment on Barge.—A Kohler 1500-watt lighting plant, type E, and a clean-up pump consisting of a 3-in. centrifugal pump powered by a Star motor are used. With few exceptions the bearings used in drives, rollers, conveyor and machinery are Timken roller bearings.

Property

The land that was secured for this operation is a "flat" along Wyandotte Creek where it debouches from the last line of foothills into the plain of the Sacramento Valley. The tract is, roughly, fan-shaped. A long, low granodiorite ridge defines the northern boundary and a gently rising slope forms the less definite southerly boundary.

Geology.—The "false" bedrock limiting the depth of this operation is the lava ash decomposed to a white pipeclay varying in thickness from 8 to 30 in., which is usual in this district. The gravel is of two "runs," or ages. The lower gravel is light blue-gray in color, called locally "blue lead" type. It is medium in size, well worn, medium sandy, fairly tight but not hard to dig. The upper gravel is red, about the same size, well worn and looser and easier to dig. Some areas have a soil capping sometimes as much as 2½ ft. deep. Depth to the "false" bedrock varies from 4 to 25 ft. and averages 12 ft. The two runs of gravel vary in their relative proportions in this depth, though the red gravel is of more uniform depth than the blue gravel.

Enrichment.—There is the usual irregularity of distribution of the gold. Channels of richer gravel indicate meandering of the streams. There is a

gradual shading out of values toward the fringes of the fan. In depth, values gradually increase to a maximum, which is generally from one to two feet above the bottom of the run. The metal content is about the same in the two runs of gravel. The gold is fine and in the red gravel is bright; in the blue gravel it is less bright and does not amalgamate quite so well. The blue gravel is not always dug. No very definite line can be drawn to decide whether it should be dug or not, but the determining factor is an economic one, roughly determined by the rate of production in ounces of gold per working hour. The actual, average depth of gravel dug during the period from April 5 to Jan. 1, 1936, was 9½ ft. Approximately one-half of the area was dredged to the false bedrock and the remainder was dredged to the blue gravel only. The deepest gravel dredged was 24 ft. There was no difficulty in cleaning bedrock, because from 6 to 12 in. of the bottom is always peeled off.

Prospecting.—The area was prospected by sinking shafts to the false bedrock. These were approximately 3 ft. wide by 6 ft. long, and were sunk by hand, using a windlass where necessary and a gasoline-motored pump when water was encountered. Lines of shafts at intervals of 500 ft. were put down across the gravel course, with spacing of about 200 ft. along these lines. There were 30 shafts on approximately 75 acres of land.

When finished a shaft would be sampled by cutting a channel 1 ft. wide by 6 in. deep in one end from the top of the shaft to the bottom. When necessary to re-sample, the channel would be cut in the opposite end. The sectional area of this channel being ½ sq. ft., the volume would be ½ cu. ft. for each foot of depth. The volume of each sample was checked by weighing before it was wet. The samples were then puddled in tubs, to break up clay lumps, and then washed in a rocker. The washed material was run through the rocker a second time before it was discarded. The rocker clean-up was panned down and amalgamated with quicksilver; the gold was extracted and weighed on an assay balance.

The shaft samples varied from 4 to 90¢ per cubic yard. Erratically high samples were thrown out or re-sampled. The average for the whole tract was 18½¢ per cubic yard. This yardage has not all been dredged yet, and the average value recovered for the period considered is higher than this figure.

For subsequent sampling a Denver mechanical gold pan was used to work down the samples. In Mr. England's opinion, these samples were higher than dredge recovery because the two crepe-rubber mats and the amalgamating pan recover more fine and coated gold than are caught on the riffles of the dredge. The object of the sampling is to show how much the plant to be used will recover in operation, and the experience of the Wyandotte company has been that a rocker or sluice box furnishes the best comparison for this purpose.

Operation

In the dredging operation the barge floats in a pond the width of the drag-line cut, 70 ft. in the work described, and the length of the barge from hopper front to outer end of the stacker—about 75 ft. (Fig. 4.)

Fig. 4.—Side view of barge with dragline dumping and stacker piling tailings. Shop in distance.

Fig. 5.—Dragline dumping bucketful of gravel into barge hopper.

The dragline digs under water from its position on the solid ground in front of the barge and empties its load of gravel into the hopper of the barge. (Fig. 5.) Thence it is washed into the trommel screen by the hopper jets and there sprayed by the jets of the screen spray pipe while being tumbled and disintegrated. Material less than ⅜-in. dia. is washed through the screen perforations into the riffle sluices under the screen.

The gold settles in the riffle spaces and its retention there is aided by periodical sprinkling of quicksilver, which with the gold gradually builds up a mat of amalgam. When the riffles are loaded, the operation is stopped and the gold is cleaned up. The minus ⅜-in. material is washed down the riffle sluices and out of the tail sluices, to drop into the pond 10 ft. behind the barge. The plus ⅜-in. material travels down the screen and discharges on to the stacker belt, which carries it out the stacker bridge to the rock pile, 30 ft. behind the barge.

Crew.—The objectives of the operation are continuous working and complete recovery of the metal content. Three shifts per day are worked. The dredge crew consists of the dragline operator, the bargeman and an oiler. In addition, on the day shift a service man maintains the supply of oil, fuel and supplies and services the dragline, and a mechanic, whose specialty is welding and torch cutting, builds up teeth and attends to special repairs and shop work. On each shift one of the men on the barge relieves the dragline operator at lunch time. In this way all operators except one have been trained on this job. Several have been sent or loaned to other outfits at various times and when expansion was necessary there have always been home-trained operators available as required.

Delays in Drag-line Operation

The necessary delays to the drag-line operation are for service, changing teeth and moving. The casual delays are for replacement of cables, for change of brake or friction band linings and for adjustments and repairs.

Service Delays.—The shovel is fueled twice a day on the day shift. With the Koehring the fuel drums were loaded on the barge and the fuel is pumped from there to the shovel by moving the barge up to it. The Lima has a mechanical pump for fueling.

Changing Teeth Delay.—The teeth are of manganese steel and are built up with the arc welder when dulled. A set is dulled and changed on every shift. These changes take from 10 to 20 min. each. This gives the welder three sets of teeth to build up every day. If the welding is well done and the teeth suffer no extraordinary wear or accident, the life may be four months or better.

Moving Delay.—Moving the dragline after finishing a cut is a very short operation if it is on solid, level ground. If mats have to be used or if much cutting, filling or leveling is necessary, it may require ½ hr. With a cut 70 ft. wide in 6-ft. ground advancing about 12 ft. on each move, there would be three moves on each shift.

Changing Bands.—The wear of the frictions and brakes must be taken up as it occurs. When worn out they must be changed for spares. Different linings are continually tested to find the kinds that cause least delays. After long use the Koehring bands were more difficult to adjust

and gave less even wear than those on the Lima. The latter has very wide bands and power control that grips with little slipping or wear.

Cable Replacements.—Cable wear is fairly regular. On the Koehring drag cables average 16,000 yd.; hoist cables, 40,000; trip cables, 10,000. Cable life on the Lima has not been determined.

Barge Delays

Necessary Delays.—The necessary delays on the barge are for servicing and cleaning up. When the shifts change the engine is stopped and inspected and serviced with oil. The fueling is done without stop. As the drag line delays longer, servicing is charged to it in the delay time.

Casual Delays.—Casual delays on the barge are for changing lines, plugging of the hopper or breakdowns in some part of the equipment. Screen trouble has been mostly in the gear-reduction box of the drive. Water-system troubles are due to air leak in the pump, clogging of the pump, intake or spray pipe with trash and belt trouble. The hopper or stacker conveyor may become clogged with large rocks or stumps and occasionally may be overloaded.

Except for extraordinary delays, such as damage to the hull, change of screen, rebuilding of screen-drive speed-reduction box and the clean-up, the barge causes a small proportion of the delays. The clean-up offers an opportunity for overhauling equipment every week or ten days.

Log

A log book is kept on the barge and the running time and delay time are entered as they occur, with notation of their cause. Changing conditions show up in the delay record. A comparison of three 10-day periods of operation in the first three months has been worked up and is shown in Table 1. "Working time" is the time the drag line is actually digging gravel. All other time is "delay time."

The first period covers the first 10 days working with this barge. In spite of delays due to breaking in and tuning up, the average digging time was over 50 per cent of the total. There was some revamping that required a shutdown for a couple of shifts. After the first month the digging improved. A better percentage of working time was obtained. The greatest delay on the dragline was for moving time as it had to move on timber mats going down the creek. Changing the mats each time and being mired down a couple of times made this average high. Considerable delay was caused by boulders during the first month, as they were hard on the buckets. After that the trouble from that cause lessened.

The barge was sunk and facilities for taking care of such cases had to be improvised. That delay has not recurred. In the last period shown in Table 1 there were two unusual repairs that show up on barge delays, which were troublesome. The gear reduction in the screen drive fur-

TABLE 1.—*Comparison of Three Ten-day Periods*
PERCENTAGE OF TOTAL TIME

	1 April 5–16	2 May 1–10	3 June 1–10
Working time	51.5	63.2	57.7
Dragline delays:			
Service	4.8	3.8	3.4
Moving	11.4	8.6	1.6
Teeth	2.7	1.6	1.9
Bands	2.7	5.2	1.3
Cables	1.6	5.8	1.5
Bucket, chains	3.6	2.8	0.5
Engine	1.0	1.2	0.1
Total dragline	27.8	29.0	10.3
Barge delays:			
Hopper	2.1	2.2	0.8
Lines	0.3	0.3	0.1
Pump	2.0	0.2	8.4
Screen	0.1	0.9	12.4
Stacker	0.6	0.7	2.1
Hull	6.6		
Engine		0.2	1.5
Clean-up	2.9	3.3	
Total barge	14.6	7.8	25.3
Shut down	6.1		6.7
	100.0	100.0	100.0

nished by the manufacturer proved unequal to the job. The bearings were too small and pulled loose and the gears wore excessively. The repairs provisionally made did not stand up because the materials in the construction were too light. Mr. England fabricated a new reduction, using Timken bearings and specially heat-treated gears that completely eliminated further trouble from this source.

The pond water became so thick from fine trash that considerable difficulty was experienced with the screen box on the pump intake. The skimmer jet could not keep it free and an extra man was used on each shift provisionally for a few shifts until a new, larger screen box was fabricated of steel angles with more screen area, which greatly helped the situation. This trouble is always most threatening when water gets low and evaporation is high.

Because of these two difficulties the last period shown in Table 1, when they were most acute, indicates an unusual proportion of delay for the barge. These two items account for 83 per cent of the barge delays. On the other hand, dragline delays were low during the same

period. The most significant improvement in this period was in dragline moving time. The creek bed was finished and the timber mats were no longer necessary.

During the entire period covered by the three stages in Table 1, the average proportion of working time to total time was 65.5 per cent, so that all three periods shown are below the average.

Tailings Losses

No very systematic check is made on losses, although tailings are panned at irregular intervals. In the coarse tailings there may be some loss due to poor washing or there may be integrated lumps, bound by adobe clay, which are too tough to disintegrate in the trommel screen. The remedy for the first condition is to improve the washing immediately, which is always done. Many times the clay lumps have been tested and occasionally some colors are found, but the loss is small.

The sand tailings may carry gold that is not held in the riffles because: (1) it is too fine to settle; (2) it will not amalgamate; (3) conditions are unfavorable to settlement of the gold because of boiling of the riffles or overloading. Testing of the sands shows there is some loss there. It is estimated not to exceed 2¢ per yard. Half of the pans show no colors, and colors that are found are usually very fine. Some few colors do not amalgamate. A trial of putting the sands over a carpet covered with expanded metal recovered some fine gold but it could not be made a practicable part of the recovery system. The conclusion is that the sand loss is mostly very fine gold.

Arrangements are being made to test the use of an impact amalgamator on an undertow cut from the tailings launder. The actual recovery from the ground so far worked has exceeded the test value slightly.

Accessory Equipment

A combined machine shop and warehouse was built on the property for the protection of equipment and supplies and facility in making repairs. It is 20 by 40 ft. framed in sections so as to be portable, of corrugated iron on two by fours. The shop has a 12-in. lathe, a drill press, an emery wheel and wire brush, work benches and welding table, gas-fired retort, bullion balance, and bins for hardware supplies. An arc-welder and oxyacetylene equipment are mounted on a truck for use wherever needed. There is a service truck for general work. A caterpillar tractor with bulldozer attachment is of wide general use for clearing, road building and heavy hauling.

Costs

Investment.—The cost of this plant is approximately as follows: dragline excavator, $18,000; barge and washing plant, $12,000; accessory

equipment, $10,000; total, $40,000. The exact costs cannot be given because of the manner of purchasing and exchanges effected. Mr. England has been instrumental in starting several similar operations, some dependent on the Wyandotte operation and others independent of it. These have permitted unusually advantageous exchanges of used equipment while benefiting ambitious men of his organization to successfully start or help in like enterprises. These experiences demonstrate

TABLE 2.—*Operating Account, Wyandotte Gold Dredging Co., April* 5, 1935 *to Jan.* 1, 1936
557,764 CUBIC YARDS

		Total	Cents per Cu. Yd.
Revenue: 3681.5 oz		$118,189.83	21.19
Less property payments		10,222.40	1.83
		$107,967.43	19.36
Expenses:			
Barge: Pay roll	$13,980.75		2.51
Maintenance	5,018.36		0.90
Oil, gas, lubricants	1,813.63		0.32
Shop electrical power	56.89		0.01
Total		20,869.63	3.74
Shovel: Pay roll	7,842.65		1.40
Maintenance	8,862.85		1.60
Oil, gas, lubrication	1,810.94		0.32
Total		18,516.44	3.32
Water		377.10	0.07
Auto and truck		698.64	0.12
Insurance: State compensation	1,695.84		
Burglar and auto	81.20		
Total		1,777.04	0.32
Quicksilver		165.63	0.03
Office		160.50	0.03
Bank charges on gold shipments		885.29	0.16
Telephone		106.71	0.02
Surveying		96.00	0.02
Prospecting: labor and supplies		1,257.50	0.22
Clearing: contract and equipment		5,496.00	0.99
Total expense		$ 50,406.48	9.04
Net operating profit		$ 57,560.95	10.32

that amortization cost of this operation will be a negligible figure. A book figure of maximum amortization charge may be as much as 2.86¢ per cubic yard. Depreciation of 25 per cent per year would seem fair, and would allow a salvage value of 50 per cent after two years use. On this basis amortization and interest would amount to 1.5¢ per cubic yard.

Operating.—The costs in Table 2 cover the period from April 5, 1935 to Jan. 1, 1936. They take account of land payments, prospecting, surveying and clearing as well as all repairs, replacements and alterations (maintenance) and all overhead expense.

The net mint returns show revenue of 21.2¢ per cubic yard. The excavation cost is 3.32¢ and barge-operation cost is 3.74¢ per cubic yard. The total operating cost is 9.04¢ per cubic yard, and includes some advance property payments and all clearing and prospecting expense for the whole tract. The profit on this basis is shown, therefore, as 10.32¢ per cubic yard.

DISCUSSION

(Ira B. Joralemon and Wilbur H. Grant presiding)

C. M. Romanowitz,* San Francisco, Calif.—Have you made any determination as to limitations on a dragline outfit?

J. F. Magee.—On depth?

C. M. Romanowitz.—Mr. England told me last year that he would work a dragline outfit on any formation that was fairly easy to dig, a formation that had a very soft bedrock.

J. F. Magee.—There are certain very definite limits as to that. The depth should not be over 20 ft. It should be in gravel that is not too coarse wash. There should not be any boulders. We prefer a soft bedrock or a false bedrock. It is all right working on decomposed granite because the bottom cleans very well, too. If the bottom is hard, undecomposed rock, it is difficult to clean it and most of the values are usually near the bottom. You will run a chance of losing good values.

C. M. Romanowitz.—I believe that a number of dragline operations now starting up will not prove to be practical ventures because of the hard formations and depths. Quite a few failures are likely with this type of equipment. I take this opportunity to compliment Mr. England and Mr. Magee on their operation.

W. W. Bradley,† San Francisco, Calif.—During the last three years there has been considerable equipment of that type idle and many inquiries were received by the State Division of Mines from the owners—former contractors and road excavators—for the location of deposits of gravel upon which they might go to work and get out the gold. These men were experienced excavators and could handle a large amount of material in a day at a very low cost per yard. However, they were not familiar with the recovery of gold from the gravel.

This operation is different from the gold dredge, which operates continuously with a continuous bucket discharge and a continuous feed to the sluices. The drag-

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† State Mineralogist of California.

line is intermittent, and if the feed to the sluices is intermittent there cannot be efficient recovery.

Then there is the matter of screens and grades, the sizes and types of sluices, the amount of washing, etc. There were those who, while they could excavate at a cheap cost, did not get the values out of the ground. It was partly by reason of not having the correct setup and partly because they did not understand gravel deposits from the standpoint of hard or soft bedrock or possibly large boulders or maybe even cemented ground.

E. W. Ellis,* Grass Valley, Calif.—A tailing recovery operation that I am interested in has been using a fixed dragline powered by a 50-hp. double-drum electric hoist. The bucket had to be pulled through tree roots and clay banks in its travel from the sand deposit to the dumping platform. This equipment was replaced by a portable, gasoline-driven dragline and trucks at considerable saving in power cost.

Power costs were based on peak load. In operating the 50-hp. hoist the operator would pull on an occasional load until the peak reading had increased. Consequently total power costs were higher than was justifiable considering work performed.

In addition to a saving in power the portable, gasoline-driven dragline afforded a more elastic system thus permitting a selectivity of the sands to be treated.

(*Gerald Sherman presiding*)

G. Sherman,† New York, N. Y.—One of the objections to the dragline is that there are certain conditions unsuited to its use. One is a submerged channel with gold on the bedrock.

C. F. Jackson,‡ Washington, D. C.—One objection, I have heard, is that with the dragline you cannot clean up the bedrock well under water, but I think it would be perfectly feasible to leave a little ground in place, maybe 5 ft. in thickness, excavate in the dry, and then later take the rib which you left and put that through the washing plant.

The author does not say what his recovery is, and there are very few placer operators who can tell you this. If we knew how much gold was lost in some of the old dredge tailings, we would possibly be horrified. The coating of particles with oxide of iron is probably one of the reasons for the losses.

R. W. Smith,§ Atlanta, Ga.—In Georgia we have one successful dragline placer operation in which the gold-recovery equipment is on runners, in three separate units, dragged forward by the dragline in a new position, when it moves. It has been in operation for three years on placer ground that had been worked in the old days with hand equipment. I have been told that the ground worked averaged about 40¢ per cubic yard, but as to what the recovery is, I do not know.

G. W. Metcalfe,|| Boston, Mass.—I would like to know whether, in almost any barge location, you could not also get a location for a small dredge and would not the dredge be the better installation? I understand that a small dredge costing about $50,000 has been developed. I think that cost would not be in excess of the dragline equipment and might work better.

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† Consulting Engineer.

‡ Chief Engineer, Mining Division, U. S. Bureau of Mines.

§ Georgia State Geologist.

|| Consulting Engineer, U. S. Smelting, Refining & Mining Co.

C. F. JACKSON.—It has been largely a matter of cost. This is a method, though, that I think is feasible in many instances. It has much to commend it. Many excavating contractors with idle equipment have turned to placer mining. They knew what it cost to dig gravel, but many of them went "on the rocks" because they forgot that they had to dispose of the tailings. With this system you dispose of the tailings in the same way as with the dredge; if you had a dredge outfit costing not more than $50,000, it would perhaps be preferable. At the operations I saw, it was hardly worth while to invest in a large dredging outfit. It is usually pretty shallow digging, about 6 to 8 ft.

G. SHERMAN.—An important part of the increased gold production has come from placer mining operations and much attention is being directed to small operations which must be handled by draglines, etc., or small dredges. Dredges are sectionalized with unit pontoons assembled for the hull so that they can be taken apart and moved to another site. This reduces the investment in plant, which is not restricted to one small alluvial deposit.

J. F. MAGEE (written discussion).—Recovery, as mentioned by Mr. Bradley and Mr. Jackson, depends on the experience of the operator just as with any type of dredging. The means of recovery are exactly the same as are standard on the large bucket-line dredges. In this newer method of excavating the experienced operators follow the practice of the older dredging-system recovery plants and obtain comparable results. Tailings are tested and losses are remedied if possible.

In one of the districts there are known losses due to the gold being closely locked in a clayey, granitic gravel. A new, large, modern bucket-line dredge has been working the same material during the same period and in spite of the use of pebble and ball milling, jigs and tables, has not obtained practical recovery over 50 per cent, which is what the dragline dredges do there.

The only reason inherent in the dragline method of dredging that may interfere with good conditions for gold recovery is the point brought out very clearly by Mr. Bradley, that is, continuous feed by the bucket-line excavator is superior to the intermittent feed of the dragline excavator in maintaining an even flow of material over the riffle sluices. This inferiority can be remedied to some extent by (1) use of an ample-sized, flat-bottomed hopper (25° slope) which requires liberal addition of water to sluice the feed down to the trommel screen and flattens off the peaks of intermittent feeding considerably, and (2) use of screw feed in the forward, blank end of the trommel screen. This device also serves as a scrubber for breaking lumps. Neither remedy is completely successful in giving as even, uniform flow of material over the riffle sluices as does the bucket-line dredge. But it is to the credit of the riffles, developed by long experience on the large dredges and used by the successful dragline dredges, that they do an efficient job of saving in spite of such conditions. The over-all efficiency of extraction of the dragline dredging methods has been proved by final results on well-tested ground in numerous examples to be over 90 per cent. That there are examples of much less efficient extraction does not reflect as much on the principles of the method as on the skill displayed in its use.

I wish to emphasize the point made by Mr. Sherman as to the mobility attained by sectionalized plants. In late 1935, with the aid of a welder-mechanic and the owner, I built an all-steel washing plant, in units, assembled on a barge of steel pontoons for dragline operation which has worked continuously since that time and has moved on to and dredged out seven different tracts of gravel. From the time dredging operations cease on one tract until they are started on full scale on another tract requires only three days. It has been a continuous source of profit for its owner and works as

efficiently now as at any time. Quite a few of the other dragline dredges have been moved two and three times in that period.

The interest displayed in this subject apparently justifies an extension of material to cover more thoroughly the limitations of dragline dredging. In this connection I present the following extract from my paper as published by the Canadian Institute of Mining and Metallurgy:

EXTRACT FROM TRANSACTIONS OF CANADIAN INSTITUTE[1]

Limitations

There are physical and economic limitations to dragline dredging that pertain both to the conditions of the deposit and the manner of working it. To be successful, conditions of the deposit should be favorable as to (1) values, (2) depth, (3) bottom, (4) character of gravel, (5) surface, and (6) extent. Unusual conditions and those unfavorable in some degree may be overcome to some extent by modification in design or method of operation.

Values

The lower limit of average value cannot be fixed. An experienced operator has a fairly definite idea of it, but it would be unwise to make a specific pronouncement, for it will vary for every condition and operation. The section on costs will give the basis of the method of approach to this fundamental problem.

The higher the values, the greater the degree of unfavorable elements that may be overcome; the lower the values, the more favorable must be the other conditions. The conditions that give rise to high values are frequently unfavorable to dredging. Coarse gold is quite likely to be associated with heavy gravel and to be very erratic in its distribution. If most of the values are on the bedrock, the distribution is likely to be within narrow limits and the gold difficult to recover. High values may not be an unmixed blessing. Gold that is well distributed in area is likely to be better distributed in depth, that is, it will not all lie on bedrock but will show from the surface down.

As amalgamation is such an important part in the recovery, it is important that the gold amalgamate well, that it be bright and free. Rusty and black gold, and gold attached to quartz particles, is difficult to save.

The most favorable conditions for values are that the gold be fairly uniform in distribution areally and show some distribution from top to bottom also, and that it should be bright and free.

Depth

While the limits of depth of gravel that can be successfully worked may be modified by adapting the equipment to special conditions, nevertheless, they are more definite than any of the other limitations imposed by this method of dredging. The lower limit is conditioned by the depth of water necessary to float the plant on the pond. Four feet does very well for the plant of usual size. Two and a half feet can be dredged with difficulty. This difficulty may be alleviated by lessening the draft of the barge by increasing its area.

The higher limit of depth ranges from 15 to 25 ft. The ordinary dragline with 1¼-yd. bucket and 50-ft. boom cleans bottom fairly well and does not lose

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[1] Dragline Dredging. Can. Min. and Met. *Bull.* (Feb., 1937) 119–125; *Trans.* C.I.M.M. (1937) **40.**

much digging efficiency at 15 ft. depth. Beyond this depth there is sacrifice of speed and of values. The larger draglines, with longer booms and more power, dig to 25 ft. without apparent loss of efficiency. The increased output of the larger dragline requires a larger washing plant, which raises the investment and requires a larger reserve of yardage to justify this increased cost. These initial costs, however, are likely to be more than offset by lower production costs. Since greater depth of gravel increases the yardage in a given area, the deeper deposits are those most likely to provide a good reserve. Thus there is a trend just now to look for deeper deposits.

Bottom

One of the important points to be considered is the nature of the bottom or bedrock. The most desirable condition is that it should peel off easily so that from six inches to a foot may be dug. Thus, several passes with the dragline bucket, digging off a few inches of bottom each time, will pretty thoroughly clean up the higher values usually found near the bottom of the gravel.

Occasionally there is a change in the gravel where the "pay" stops, and the older, basement gravel is likely to be tighter and harder to dig. Under such conditions, the transition is noticeable and yet some of the lower gravel can be peeled off to ensure cleaning up the bottom. The transition is usually marked also by change in color of the gravel. In the absence of such features, the transition may be difficult to recognize. However, experience teaches, and the dragline operators acquire an instinctive "feel" for the "bedrock."

A distinct transition, or a definite marker for the bottom, is a considerable advantage. It gives definite evidence that all the pay gravel is being dug, which is always reassuring information. Probably lava-ash or tufa, which decompose to kaolin in moist conditions, furnish the best bedrock to dig to. As long as thin slices are peeled off, it works very well. When enough is dug to form large balls, these are troublesome.

Granite, also, decomposes to a soft bedrock under cover of gravel. Diorite and andesite do not decompose so easily and are more likely to give difficulty. Dykes of trap will generally stand above the surrounding bedrock; they are difficult to cross and they usually trap values in such a manner that they are particularly difficult to recover. Serpentine is often hard, though it will decompose below gravel under favorable conditions.

Shale is very often soft and is satisfactory bedrock if it lies flat. Steeply pitching shale bedrock is difficult to clean and very troublesome to work with if not soft. The more slaty the shale, the less decomposition or softening takes place. Sandstone is usually soft and a good bedrock. Any of these rocks may be hard where they are exposed in a stream bed and thus are subject to constant erosion that removes the softened material as fast as it is formed.

Character of the Gravel

The gravel may be fine or coarse; worn or angular; sandy or clayey; loose or tight; even or uneven. As to size, the most favorable gravel is medium. Boulders over 18 in. in diameter become troublesome to handle and very coarse gravel is difficult to dig. There is some relation between coarse gravel and coarse gold though it may not be as definite as popularly supposed. There is a similar relation between fine gravel and fine gold. Thus fine gravel may be undesirable, not from the digging standpoint but from its relation to fine gold.

Worn gravel has traveled far enough to free all gold from attached pieces of quartz. The gold is free and polished unless subsequently coated. It is likely to be fairly uniformly distributed. But in angular gravel, all the gold has not been freed from

quartz particles, nor has it been polished and distributed through the mass of the gravel. Such angular gravel is always suspect and requires close testing and inspection.

In general, a sandy gravel is a loose gravel and a clayey gravel is tight. A gravel that is too loose does not hold up the gold, but such a gravel is easy to dig and easy to wash. Therefore it is possible to work a sandy gravel of lower value than a more clayey one, because the latter is harder to dig and harder to wash.

The uneven qualities of gravel that may increase the difficulties of digging and washing it are layers or bars of soil, clay, sand, or hardpan.

Surface

The surface and topography of the gravel deposit influence the possibility of dredging and the cost of the operation. The surface may be flat and smooth, or rolling, broken, or sloping. It may be clear, wooded, or brushy. The most desirable surface is a flat, smooth one without brush or timber. The operation may be carried on in rolling or broken terrane at not much sacrifice of efficiency if the pond can be carried across the irregularities without extra work. With the aid of a tractor with bulldozer, the road can be leveled for the dragline, and dams can be pushed up across basins or depressions where the ground is too uneven for the dragline conveniently to do this work.

The dredge can ascend a 3 per cent slope without much difficulty and can descend a slope of 2 per cent with about the same effort. Steeper grades can be managed in an emergency but at greater cost and some sacrifice of efficiency.

The clearing of wooded land may cost as much as $100 per acre for densely wooded, heavy timber. This constitutes a heavy charge on a shallow deposit. Cleared land has quite an advantage.

Extent

The deposit or deposits must have sufficient extent so that their yield will pay off the investment and allow a profit. A volume of 400,000 yd. is a minimum often quoted by experienced operators for ordinary gravel and ordinary plant and equipment. Estimation of the size of deposit necessary in any particular case presents a simple problem, whose solution requires experience and an ample safety factor rather than great accuracy of calculation. Naturally, conditions are most favorable where the deposit is of large extent, because this allows a wider base for investment and initial outlay, which in turn reduces the operating costs and facilitates the operation. The longer life assured by a large reserve of gravel permits more opportunity for organizing the operation in complete detail.

Dragline dredging has one outstanding advantage in that the plant may be built easily portable. It can then be moved onto a very small piece of ground, for the moving expense is small. Provided there is sufficient yardage to pay off the investment in the plant, this yardage may be divided in areas of very few acres in each, provided these areas are not too widely separated.

Summary of Limits

An average plant, such as the Wyandotte plant, may be built and operated with average expectancy of success on a deposit of placer gravel that has recoverable values double the operating expense, is between 4 and 15 ft. in depth, with favorable bedrock, no boulders, not more than 20 per cent soil or clay seams, with surface not too rough nor heavily timbered, and that has not less than 400,000 yd. of gravel, which may be in several separate, but nearly adjacent, tracts. This general specification summarizes the limits much more definitely than is required, in order to indicate clearly that, within these limits, the dragline dredge method of operation is the most adaptable, adequate, and economical.

Costs

The fundamental limitation of any operation is the economic one of costs. Every other limitation is finally fixed by its effect on the cost and subsequent profit. In order to indicate how the variation of different conditions influences the costs of operation, a comparison of two plants identical in size and design to that of the Wyandotte, but working deposits distinctly different, is given in Tables 2 and 3. These plants are here designated Plant *A* and Plant *B*.

The dredge of Plant *A* is identical in practically every detail with the Wyandotte dredge. The dragline is a P. & H. 702 with Atlas Diesel power and 1¼-yd. bucket on a 50-ft. boom. But the gravel differs in several ways. The values recovered are only slightly higher, just under 23¢ per cubic yard. The average depth is about the same, 9½ ft. The bedrock is decomposed granite, which digs very well but is hard to distinguish from the gravel because the latter contains so much of the same material. Thus the character of the deposit is quite different from that of the Wyandotte.

TABLE 2.—*Operating Account, Plant A*
April 12th, 1935, to April 23, 1936
(458,000 cubic yards)

		Total	Cents per Cu. Yd.
Revenue: 3,557.05 oz. gold		$104,781.31	22.88
Less royalty payments		12,475.00	2.73
		92,306.31	20.15
Expense:			
Barge:	Pay roll	$ 19,353.45	4.24
	Maintenance	9,715.92	2.12
	Fuel, oil, supplies	1,568.78	0.34
	Total, barge expense	$ 30,638.15	6.70
Shovel:	Pay roll	$ 7,627.23	1.67
	Maintenance	4,815.32	1.05
	Fuel, oil, supplies	4,809.73	1.05
	Total, shovel expense	$ 17,252.28	3.77
General:	Prospecting	$ 810.50	0.18
	Quicksilver	241.21	0.05
	Other equipment expense	1,551.43	0.34
	Mileage	405.75	0.09
	Insurance	1,711.52	0.37
	Water	1,020.50	0.22
	Office	324.64	0.07
	Telephone	123.56	0.03
	Interest	213.03	0.05
	Total, general expense	$ 6,402.14	1.40
Total Expense		$ 54,292.57	11.85
Net Operating Profit		$ 38,013.74	8.30

From 60 to 70 per cent passes through the trommel screen. The gravel is very much harder to dig and harder to wash than at the Wyandotte. Also, both the digging capacity and the washing capacity are less, and the operating costs are, consequently, higher.

Plant *B* has exactly the same size of washing plant and the same size of dragline as Plant *A*, but a 1½-yd. bucket with 50-ft. boom is used. The values are lower and the gravel is shallower, 6 ft. in depth. The bedrock is good and easily distinguished and the character of the gravel is exceptionally favorable for digging and washing. It is loose and sandy with a high proportion of coarse gravel, 60 to 75 per cent, that does not pass through the ⅜-in. openings of the screen. There is some clay, but it is seldom troublesome.

TABLE 3.—*Operating Account, Plant B—Two Dredge Plants*
AVERAGE 4½ MONTHS' OPERATION TO SEPTEMBER 1, 1936
(689,820 cubic yards)

		Total	Cents Per Cu. Yd.
Revenue: 3,414.10 oz. gold		$98,870.67	14.33
Less property and royalty payments		6,858.57	1.00
		$92,012.10	13.33
Expense:			
Barge:	Pay roll	$14,914.80	2.16
	Maintenance	6,402.95	0.93
	Fuel, oil, supplies	2,568.88	0.37
	Total, barge expense	$23,916.63	3.46
Shovel:	Pay roll	$ 9,838.40	1.43
	Maintenance	3,961.94	0.57
	Fuel, oil, supplies	2,725.86	0.40
	Total, shovel expense	$16,526.20	2.40
General:	Water	$ 2,546.00	0.37
	Quicksilver	53.46	0.01
	Insurance	985.76	0.14
	Office	1,021.60	0.15
	Total, general expense	$ 4,606.82	0.67
Total Expense		$45,049.65	6.53
Net Operating Profit		46,962.45	6.80
Less depreciation, 33⅓% per annum		7,500.00	1.08
		$39,462.45	5.72

NOTE: Depletion not accounted; expense includes salaries of partners and all maintenance charges.

Present Status of Hydraulic-mine Debris Disposal in California

By Walter W. Bradley,* Member A.I.M.E.

(San Francisco Meeting, October, 1935; New York Meeting, February, 1936)

Mining by hydraulic process of the important gold-bearing gravels of the Sacramento Valley in the basins of the Yuba, Bear and American rivers began in 1853, and continued at an ever-increasing rate for 30 years. In its most active days it reached an annual production of $15,000,000, and averaged $10,000,000 per year over the 30 years. During that period no provisions were made to prevent the debris from being discharged into the water courses, and ultimately large quantities of silt, sand and gravel were washed down to the lower valleys; filling channels, causing overflow of agricultural lands and difficulty in maintaining navigable depths in the Sacramento River.

Agitation and suits filed because of these damages finally resulted in the injunction decision of Judge L. B. Sawyer,[1] in the U.S. Circuit Court, Jan. 23, 1884, which closed down practically all hydraulic mining operations not having means to restrain their debris. In 1893 (approved March 1, 1893), Congress passed the act introduced by Representative Anthony Caminetti, of California, creating the California Debris Commission, composed of three officers assigned from the Engineer Corps of the U.S. Army, and since that time it has been legal and possible within the drainage systems of Sacramento and San Joaquin rivers to conduct hydraulic mining of gold-bearing gravels behind dams built under this commission's permit and supervision.

In the watersheds of the Trinity and Klamath rivers, which flow directly to the Pacific Ocean and do not affect navigation or agriculture, operating permits from the California Debris Commission are not required. There is a measure of restriction there, however, through what is known as the Trinity and Klamath River Fish and Game District. Chap. 215, Stat. 1931, in order to secure relatively clear water for the fishing season, provides that between July 15 and October 15 each year there must not be discharged into these rivers debris that will cause a turbidity greater than a content of 50 parts per million in the

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[1] Edwards Woodruff v. North Bloomfield Gravel Mining Co. et al., 18 Fed. 753.

water at a point one mile below the junction of Trinity River with its South Fork, and one mile below the junction of Scott River with the Klamath.

In Sec. 23 of the Caminetti Act, authorization was given, and it was assumed that provision of funds would be made from time to time, for construction of debris dams by the Commission, the cost of which would be repaid by storage charges by the mines operating above those dams. This Sec. 23 provides in part, as follows:

> That upon the construction by the said commission of dams or other works for the detention of debris from hydraulic mines and the issuing of the order provided for by this act to any individual, company, or corporation to work any mine or mines by hydraulic process, the individual, company, or corporation operating thereunder working any mine or mines by hydraulic process, the debris from which flows into or is in whole or in part restrained by such dams or other works erected by said commission, shall pay a tax of three per centum on the gross proceeds of his, their, or its mine so worked. . . . All sums of money paid into the treasury under this section shall be set apart and credited to a fund to be known as the "Debris Fund," and shall be expended by said commission under the supervision of the chief of engineers and direction of the Secretary of War, in addition to the appropriations made by law in the construction and maintenance of such restraining works and settling reservoirs as may be proper and necessary.

Thus far, however, in all of the 42 years since the enactment of the Caminetti Act, no appropriation has been made for the building of such debris dams. Due largely to the difficulty and cost of providing the required storage works, except by a few scattered mines, operations have been greatly restricted. At the 1934 session of Congress, Representative Harry L. Englebright, of California, introduced and succeeded in having passed an amendment (approved June 19, 1934) to Sec. 23 of the Caminetti Act providing for a more equitable and feasible method for repayment of dam-construction costs through the storage charges.

This amendment provides that mines storing their debris behind any dams or other works erected by the commission shall pay a tax for each cubic yard mined as measured in the natural bank, equal to the total capital cost of the dam, reservoir and rights of way (not including interest), divided by the total capacity of the reservoir for the restraint of debris, and that revenue from the sale of this storage shall be set aside and credited to a fund to be used for repayment of any funds advanced by the Federal Government or other agencies for the construction of restraining works and settling reservoirs, and for maintenance.

Pursuant to a resolution adopted Aug. 18, 1933, by the Committee on Rivers and Harbors of the House of Representatives, the Board of Engineers for Rivers and Harbors was requested to review the reports on Sacramento River on the subject of debris control, with particular reference to the construction of certain dams and the participation or

the interest of the Federal Government in their construction. The job was delegated, in turn, to the California Debris Commission.

After working for approximately 1½ years, and the expenditure of $95,000, the Commission rendered its report under date of Feb. 13, 1935, to the Board of Engineers for Rivers and Harbors, and while finding that dams at certain specified points were feasible, physically, drew certain conclusions therein from economic and federal-interest standpoints, which were considered by interested Californians to be not in accordance with the actual facts. Particular objection was raised to the statement of "insufficient information available to accurately determine the gold content of the gravels."

A hearing in protest of those conclusions was arranged for and held before the Board for Rivers and Harbors in Washington, on April 15, 1935. Besides Congressman Englebright and the representatives of a number of gravel-mine owners, the State Mineralogist of California, by direction of His Excellency, Governor Frank F. Merriam, appeared before the Board and presented both briefs and oral arguments. As a result of that hearing and of study of the reports and data relating thereto, the Board of Engineers for Rivers and Harbors, under date of May 23, 1935, rendered a report in which it reversed the conclusions of the California Debris Commission, and stated:

> After careful consideration of all available data and all factors concerned, the Board concludes that construction of four storage developments essentially as planned by the California Debris Commission is economically justified and that revenue from the sale of debris storage rights will reimburse the United States for the funds advanced. . . . .
>
> The Board recommends the construction, at an estimated cost of $6,945,000, of the four debris storage reservoirs in the Sacramento Basin (Upper Narrows, Dog Bar, North Fork, and Lower Ruck-a-Chucky) set forth in the plans of the California Debris Commission, with such modifications as may be approved by the Chief of Engineers, subject to the condition that work shall not be begun on any reservoir until the repayment of the capital cost thereof by the payment of taxes on material hydraulically mined from the natural bank in accordance with the provisions of Section 23 of the act to create the California Debris Commission and regulate hydraulic mining in the State of California, as amended by an act approved June 19, 1934, is assured to the satisfaction of the Secretary of War through guaranties of responsible individuals, companies and/or corporations that they will if the reservoir is constructed, hydraulically mine under license from the California Debris Commission, material in amounts aggregating the total capacity of the reservoir for the restraint of debris.

Following the presentation of the aforementioned report, there was included by amendment in the Senate, the item of $6,945,000 to the Rivers and Harbors Bill. The amendment was accepted by the Conference Committee and the House, and the bill was signed by President Roosevelt on Aug. 31, 1935. These four hydraulic-mining dams are thus given the status of an authorized federal project, with an authorization of the foregoing amount for their construction. Although the money has not

yet been actually appropriated or set aside for this purpose, it is now anticipated that that final and concluding act will soon be recorded.

The Legislature of California as its session of 1933 passed a bill (S.B. 480) by Senator Jerrold A. Seawell, of Placer County, designated "an act to provide for the organization, operation, financing, government and dissolution of placer-mining districts." It was approved by Governor James Rolph, Jr., June 12, 1933 (Chap. 899, Stat. 1933). It provides for the formation of such districts in much the same manner as already has been in force for many years for irrigation and reclamation districts in California. The board of directors of such a placer-mining district when formed is authorized among other things (Sec. 73):

73.4 To enter upon any land and make surveys, locate the necessary works and the lines for canals, and the necessary branches for the same on any land which may be best for such location; lay out the necessary canals or other means of conducting tailings to storage basins or reservoirs and to lay out such storage basins or reservoirs.

73.5 To acquire by purchase, lease, contract, condemnation, or other legal means, all lands, waters, water rights, and other property necessary for the construction, use, supply, maintenance, repair and improvement of such canals or other means used for the conduct of water and tailings and for the supply of water, and for such purpose to acquire and hold the stock of corporations, domestic or foreign, owning water, water rights, canals, water works, franchises, concessions or rights.

73.6 To make and perform agreements with any government or governmental authority, political subdivision, public or municipal corporation, for the joint acquisition, construction, leasing, ownership, disposition, use, management, maintenance; repair or operation of any rights, works or other property of a kind which might lawfully be acquired or owned by a placer mining district.

73.7 To acquire the right to store water in any basin or reservoir or to carry water through any canal, ditch or conduit not owned or controlled by the district.

73.8 To grant to any owner or lessee of the right to the use of any water, the right to store such water in any reservoir of the district or to carry such water through any canal, ditch or conduit of the district.

73.9 To exchange property or rights of the district for property or rights of any person, firm, corporation, district, or other public or municipal corporation, political subdivision, or the State.

73.10 To make all agreements and do all acts necessary to make available to land in the district the facilities for placer mining operations and the storage of tailings therefrom.

73.11 To construct dams, canals, ditches, or other means for the conducting of water and tailings from any property in the district to a storage basin.

73.12 To make any agreements or conveyances, sue and be sued, and expend the moneys of the district for the purpose of the exercise of any of the powers granted by this act.

73.13 To establish equitable by-laws, rules and regulations and to make charges for the distribution and use of water and the facilities of the district among the owners of the land. Such by-laws, rules and regulations must be printed in convenient form and distributed to such owners. To establish the general fund and any desirable or necessary special funds of the district, prescribe the keeping of accounts thereof and to require a report to the board of directors at such periods as the board may consider convenient of the finances of the district.

Sec. 74. The use of all water or other facilities required for the working of the mines in the lands of any district formed under the provisions of this act, together with the facilities auxiliary thereto and all other property required in fully carrying out the provisions of this act is hereby declared to be a public use, subject to the regulation and control of the State in the manner prescribed by law.

It is now possible, through the agency of the foregoing statute, for the owners of gold-bearing gravels in a given watershed, and minable by hydraulic method, to combine their interests and pool their resources and thus obtain water supplies, tailings storage and other facilities for the working of their several properties, either individually or jointly. Since the passage of this statute the Eagle Rock district has been organized and the bonds voted to take care of the Omega and Relief Hill deposits in Nevada County. A number of others are in the preliminary stages, but all are now marking time waiting for the Federal Government's program on the four dams authorized under the Rivers and Harbors Bill. When the government dams are built, it will be necessary for the mine owners to organize into districts under this act in order to condemn rights of way to flow water to their mines and the tailings to the storage basins.

TABLE 1.—*Sites of Debris-storage Reservoirs in California*

River	Reservoir	Gross Water Capacity, Cu. Yd.	20-year Mining Operations, Cu. Yd.
Yuba........................	Upper Narrows	118,000,000	118,000,000
Bear........................	Dog Bar	25,900,000	26,000,000
American, north fork..........	North Fork	25,100,000	26,000,000
American, middle fork.........	Lower Ruck-a-Chucky	24,900,000	24,000,000
Total.....................		193,900,000	194,000,000

There were in operation during the season of available water in 1934, five producing mines worked behind dams under permit and approval of the California Debris Commission. These were scattered through the counties of Amador, Nevada, Placer, Sierra. At the same time there were six operators in Trinity, Humboldt and Siskiyou counties. Since the formation of the Debris Commission in 1893, "1152 applications for license have been received and 833 licenses issued. About 23½ million cubic yards of gravel have been mined under these licenses. At present there are 72 licenses in force."[2]

In the course of the field work for the aforementioned report, the engineers of the Debris Commission investigated 20 sites for debris-storage reservoirs, of which six were rejected either due to foundation

[2] Report of California Debris Commission to Chief of Engineers, U.S. Army, under date of Feb. 13, 1935.

conditions or as not well adapted for the purpose. Of the remaining sites, four were selected by the commission as the most economical and best situated to provide the storage required under the 20-year plan of

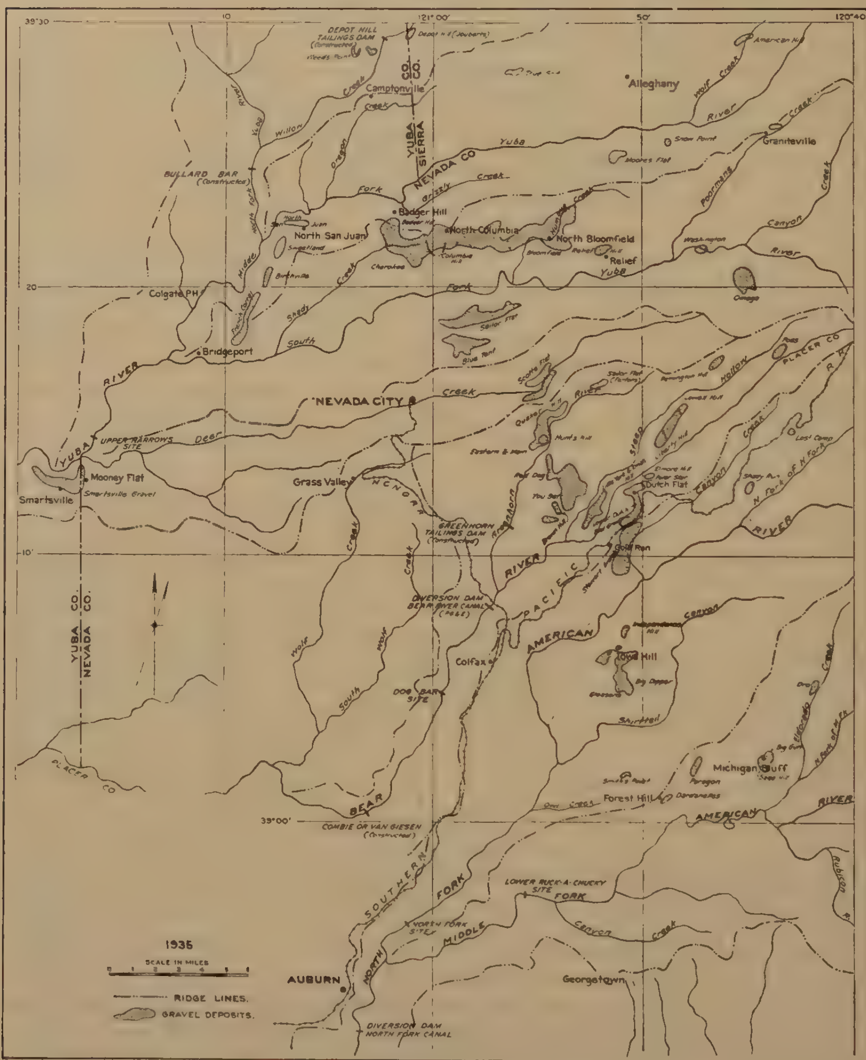

FIG. 1.—LOCATIONS OF SITES FOR DEBRIS-STORAGE RESERVOIRS IN CALIFORNIA.

operations. The capacities of these four are shown in Table 1. Their locations are shown on the map, Fig. 1.

Based upon their field examinations and other factors affecting hydraulic mining, the Commission made an estimate of the quantities of workable gold-bearing gravels and the potential mining activities to

TABLE 2.—*Commission's Estimate to Cover 20-year Period*

River Basin	Workable Gravels, Cu. Yd.	Gravels to be Mined under 20-year Plan, Cu. Yd.
Yuba, middle fork	320,000,000	60,000,000
Yuba, south fork	68,000,000	58,000,000
Bear	53,000,000	26,000,000
American, north fork	87,000,000	26,000,000
American, middle fork	76,000,000	24,000,000
Total	604,000,000	194,000,000

cover a 20-year period, which is given in Table 2. The Jarman report[3] gives the following estimates of workable gold-bearing gravels in the same areas:

	Million Cu. Yd.		Million Cu. Yd.
Yuba River, middle fork	442	American River, north fork	95
Yuba River, south fork	94	American River, middle fork	21
Bear River	33	Total	685

In addition to the several dams built and maintained by individual companies or operators for the storage of their hydraulic-mine tailings, there are two others that are available to their respective tributary areas:

1. On the North Fork of Yuba River at Bullard's Bar, 3½ miles above the mouth of the fork, a concrete dam 175 feet high was constructed by the Yuba River Power Co. in 1922–1924. Debris storage in the reservoir has been authorized by the California Debris Commission to the extent of 40,000,000 cu. yd. The present owners, Pacific Gas and Electric Co., charge for storage 2¢ per cu. yd. mined; but prior to 1932 it was 3¢ per cu. yd. About 1,650,000 cu. yd. has thus far actually been mined for storage in this reservoir; and there are 11 mines licensed to use the reservoir debris storage now contracted for amounting to about 6,000,000 cu. yd.

2. In 1928, the Combie Dam was constructed on Bear River, 3½ miles west of Clipper Gap, by the Nevada Irrigation District. Though built primarily for the diversion of water for irrigation purposes, debris storage has been sold to mining interests on Bear River, with a total of 5,000,000 cu. yd. approved by the Debris Commission. The charge for storage is 3¢ per cu. yd. It is stated that with additions to this dam, at a relatively small cost, a total of about 15,000,000 cu. yd. of debris

[3] Report of the Hydraulic Mining Commission upon the feasibility of the Resumption of Hydraulic Mining in California. A report to the Legislature of 1927. Reprinted in January, 1927, Chapter of State Mineralogist's Report.

storage could be provided. About 600,000 cu. yd. has thus far been mined for storage there.

As the storage in both the Bullards Bar and Combie reservoirs is considered to be permanent, a license to operate by the hydraulic process is granted by the California Debris Commission to applicants who own properties above these reservoirs, if evidence of authority to store debris in them is furnished.

With the construction of the four dams authorized by Congressional enactment imminent, and the utilization of the benefits and provisions of California's Placer Mining District Act, the stage is now set for increased gold production in the Sierra Nevada by hydraulic mining. This means not only the employment of many thousands of men, directly, but the rehabilitation of many towns and districts long dormant.

DISCUSSION

(*Gerald Sherman presiding*)

G. Sherman,* New York, N. Y.—Anyone who has traveled up the Sacramento Valley will realize what the tailings from old placer mines have done to the valley and will continue to do, because they are still moving slowly downstream. This new plan is a development of the Caminetti Act under which restraining dams can be built. It appears to be a more permanent solution of the problem and will permit more hydraulic mining to be done than for many years past.

H. A. Franke,† San Francisco, Calif.—This bill provides that the Federal Government will supply the money to erect four debris-storage reservoirs, the cost of which will be repaid by the hydraulic miners according to the amount of debris they discharge back of the dams.

G. Sherman.—I presume it will have to be proved and demonstrated that there is enough profitable gravel there to fill the dams.

H. A. Franke.—Evidence given at the April 1935 hearing in Washington proved that there was sufficient auriferous gravel above the proposed damsites. The 1927 report by Arthur Jarman to the California Legislature gives the amount of gold-bearing gravel available for hydraulicking.

G. W. Metcalfe,‡ Boston, Mass.—I would like to know if, at the present time, there is litigation in California in reference to irrigation districts below the hydraulic operations now in progress and if the building of these dams would in any way give the operators any protection against litigation.

H. A. Franke.—No doubt there are a few cases of litigation. Only a year ago a bathing resort complained about a hydraulic mining operation which was muddying the stream. This operator had only a brush dam for debris storage. However, there are seldom complaints against hydraulic mining where the dams are of good construction such as the Bullard's Bar Dam. The California Debris Commission (U. S. Army Engineers) requires one to secure a permit before hydraulic operations may be started.

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* Consulting Engineer.

† Mining Engineer, California State Division of Mines.

‡ Consulting Engineer, U. S. Smelting, Refining & Mining Co.

G. W. METCALFE.—I have had no experience in such litigation, but I have had considerable experience in smelter smoke litigation, which, in a way, is somewhat like it. The farmers complain that any calamity that happens to their crops is due to smelter smoke, and perhaps the farmers below the scene of operations would take much the same attitude, claiming that the very fine material not settled out would damage the farms, etc. And in considering the California situation it occurred to me that government approval of dams or construction of dams might not or could not give real legal protection, but that it might cause a presumption in the courts asked to pass on it that damage did not occur, and through such legislative action throw the burden of proof on the farmer.

H. A. FRANKE.—Such might be the case, but at present farmers are not complaining. In fact, a lot of them have turned to gold mining. However, I have not closely followed the court cases on this matter. There has been some litigation, and in the bathing resort episode I think the complainants won their case against the hydraulic interests.

G. W. METCALFE.—I would like to know how they protect the farm land.

H. A. FRANKE.—The silt from hydraulic operations eventually settles out, but a well-built dam will stop most of this silt near the mining operation and by the time the water reaches the farmer much farther downstream it will be clear. Inasmuch as hydraulic mining debris prematurely fills up storage space behind dams the present irrigation and power companies permitting this make a charge based on the number of cubic yards of gravel washed.

G. W. METCALFE.—I suppose that after they are filled up, they will be of no use for protection.

H. A. FRANKE.—A new dam would have to be constructed after one was filled with debris if more auriferous gravel is available for hydraulicking. However, the proposed dams will store a tremendous amount of debris.

G. SHERMAN.—The small rivers where the dams are situated are not navigable.

H. A. FRANKE.—That is correct. There is little navigation on the Sacramento river above the city of Sacramento. It is now necessary for the U. S. Army Engineers to dredge the river in the vicinity of the city of Sacramento because of debris due to natural erosion.

G. SHERMAN.—In the reclamation dams, like the Roosevelt Dam in Arizona, silt builds up rapidly where the stream current is lost in the lake, and can be seen and measured at the low-water stage. Material of that size drops quickly. The water clarifies itself still further below until what goes over the dam is impalpable. Nothing is lost over the dam that would damage farm lands. The mud from the Nile acts as a fertilizer, and has maintained the fertility of that valley for ages.

H. A. FRANKE.—That is also my impression. Some miners contend that the silt is beneficial rather than detrimental to the farmers, but the farmers do not see it that way.

A. MEYER.—Some time ago I did some hydraulic mining in the Chocó in Colombia, and we had quite a little litigation on account of farming land. The river there was used as a highway, and many people used canoes for carrying their products to the various villages, and the debris was piled up on each side of the river, creating a channel. Every once in a while the canoes would upset and occasionally there would be

some platinum and gold in the canoes. But here is a case where the navigation was really interfered with, and besides that there was no fishing in the rivers.

G. Sherman.—No one would challenge the statement that hydraulic mining damaged the Sacramento Valley farms, and it will be a long time before the tailings or debris will stop moving downstream, but this fact was not realized until too late.

W. W. Bradley (written discussion).—So far as clarity of the water below the retaining dams is concerned, the concrete dams designed by the California Debris Commission and the lake created above each will have such settlement capacity and area that the water passing over to the channel below will have practically no turbidity. This has been the experience with the Bullard's Bar dam. The Debris Commission will not grant permits for mining of yardage in excess of the retaining capacity of the dams. Any further mining by hydraulic process will require additional dams.

The fact that these dams are built by the U. S. Government, while not precluding the possibility of injunction suits by farmers or others below, will at least give sufficient assurance of their safety and adequacy that the likelihood of suits will be minimized.

Factors Related to Man-hour Studies in Metal-mining Operations

By George B. Holderer,* Member A.I.M.E.

(New York Meeting, February, 1935)

The relation between man-hours of labor and production may be correlated for any industry, and already it has been widely used in piecework studies. It is not in general use as yet for recording labor efficiency of an enterprise or an industry. An exception to this is the Cement Institute, which tabulates the man-hours used in each operation and

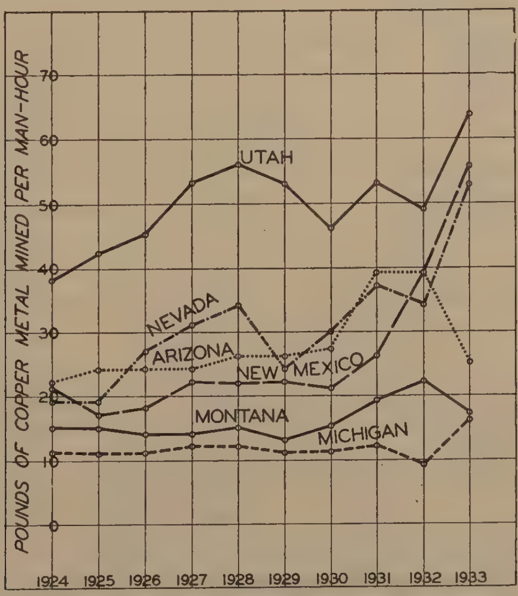

Fig. 1.—Labor productivity in copper mining.

thus gets the total man-hours per barrel of cement. The writer knows of one steel company that keeps a complete record of man-hours for every operation from mining the ore to finishing the product. No doubt many others find the information useful, and such a study could be of value to the mineral industries.

Manuscript received at the office of the Institute March 14, 1935.

* Mining Engineer, Scarsdale, N.Y.

Fig. 2.—Relative productivity of labor in various states (triangles) and irreducible mining and milling cost (excluding overhead, depreciation and depletion), in cents per pound of copper (circles). Credit (cents per pound of copper) from secondary metals, gold, silver, zinc and lead, which can be applied against the cost of copper (squares).

There is little published information available for any given industry. The Department of Labor[1] publishes a monthly bulletin with index figures on employment and wages for the main industrial groups. This is of value chiefly in showing the fluctuation in employment. The Department of Commerce[2] publishes a monthly bulletin giving figures on production, shipments, consumption, stocks on hand, and prices for every important commodity. This information presents a current survey of industrial activity as a whole. The Bureau of Mines[3] publishes an annual report giving number of men employed, man-hours worked, and production for the different divisions of metal mining, but its information is very limited. It is from this source that most of the data were obtained for the graphs used in this paper. Unfortunately the information on metallurgical plants is not broken down to show the kind of material treated.

This paper will be confined to the application of man-hour studies in metal mining. The importance of such studies and results in metal mining for 1929 is given in Bureau of Mines *Information Circular* 6503, Methods and Costs of Mining at the Metal Mines of the United States, by C. W. Wright, published in 1931.

The study of man-hours and their relation to production should be undertaken from a sectional point of view in order that a comparison of competitive factors may be made between various districts.

In the extraction of minerals there are certain fixed factors about which nothing can be done, such as location, type, size and grade of the deposit. The operating factors, such as management, labor and degree of mechanization, are variable and subject to control. The sum total of all the factors will decide whether the enterprise can compete with others of a similar nature in the same or different districts.

The man-hour is the basic unit of measurement in employment and production studies. It is a combination of the number of men employed with the number of days and hours worked, and simplifies calculations where these units are used. There will be wide variations in the output per man-hour in the same district perhaps and most certainly between different districts. In combination with other units, trends may be charted to show effects of mechanization and technological improvements, loss by idle time, the degree to which work has been spread, and the effect on the final cost of a product by a change in wages. A basis may also be established for estimating the degree of permanent employment afforded by any given industry.

For comparison between districts some of the accompanying graphs may be of interest.

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[1] U.S. Department of Labor: *Trend of Employment.*

[2] U.S. Department of Commerce: *Survey of Current Business.*

[3] U.S. Bureau of Mines: Accidents in Metal Mines in the United States.

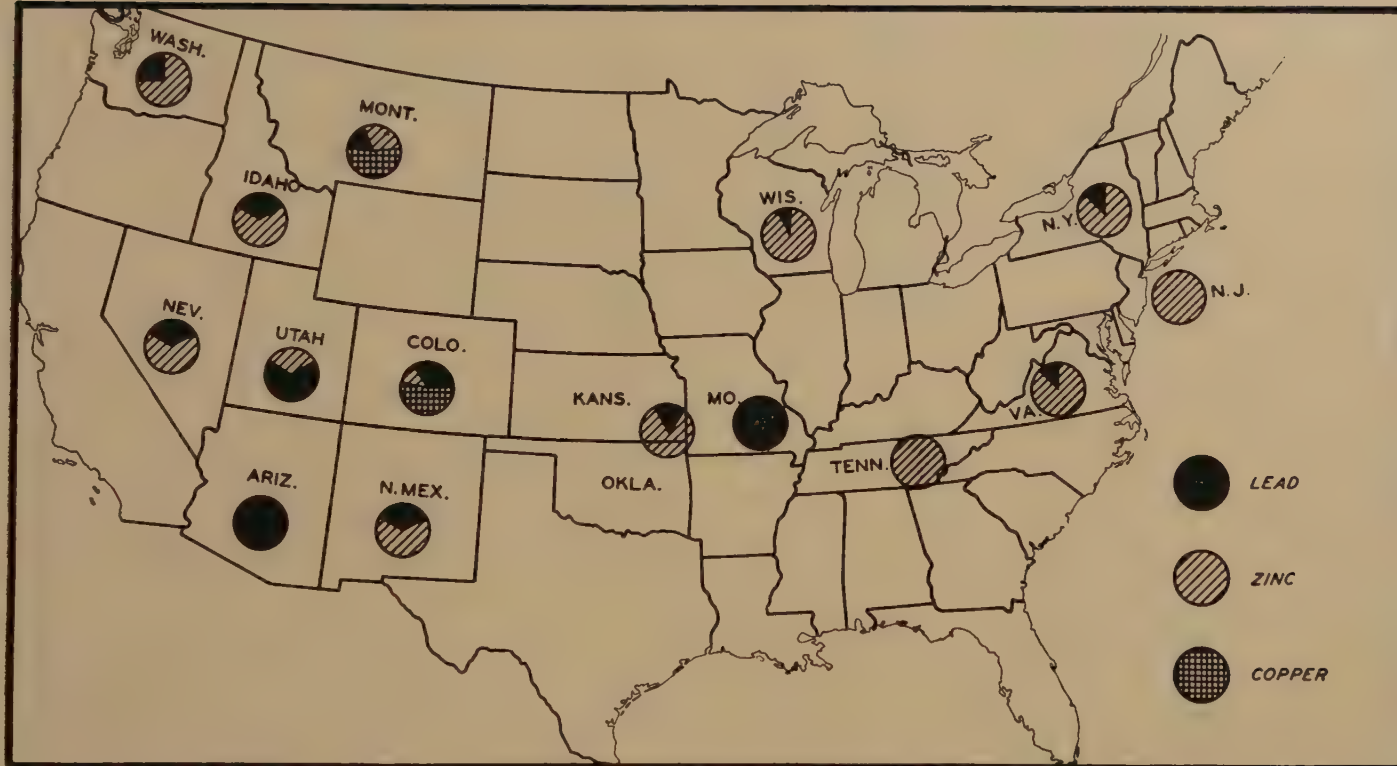

FIG. 3.—PRINCIPAL LEAD AND ZINC-PRODUCING STATES, WITH PRODUCTION FOR 1933.

Copper production and value of gold and silver (gold, $25.50 per ounce; silver, 34¢ per ounce) are indicated only when these metals are obtained in mining lead and zinc, and do not indicate the total production of each within the state. Southeast Missouri is the only large producing area where lead alone is mined without zinc or other byproduct metals. See also Table 1. Data from U. S. Bureau of Mines.

Fig. 1 shows the pounds of copper mined per man-hour by mine labor. The wide difference in productivity between Michigan and Utah, for example, brings to mind at once the different types of deposits and the mode of working them. In order to complete the picture on the competitive position of the various states, the cost of mining and milling, and credit for sundry metals should be shown as in Fig. 2. Attention is directed to low labor productivity and high cost per pound of copper and absence of credit for secondary metals in Michigan, and the reverse condition in the southwestern states.

Lead and zinc are more widely distributed than copper and present greater difficulty in charting the output per man-hour on account of the complexity of the ores. Fig. 3 shows the principal producing regions. This shows production of lead and zinc only and is not intended to show the complexity of the ores, except in two cases where it seemed advisable to include copper-production figures. Lead and zinc tonnages are shown in Table 1.

TABLE 1.—*Production of Metals for* 1933

	Lead, Tons	Zinc, Tons	Copper, Tons	Gold and Silver, Dollar
Arizona	1,690			
Colorado	2,402	1,285	4,972	$4,200,000
Idaho	74,375	21,090		1,900,000[a]
Kansas	6,089	40,947		
Missouri		5,043		
Missouri S. E.	84,980			
Montana	6,700	20,750	32,700	1,700,000
Nevada	2,320	6,150		
New Jersey		75,125		
New Mexico	11,043	30,924		
New York	1,775	17,733		
Oklahoma	18,038	91,065		
Tennessee		18,200		
Utah	58,688	29,745		2,500,000
Virginia	1,341	14,570		
Washington	840	3,369		
Wisconsin	540	7,800		
Tri-State total	24,127	137,054		

[a] Silver only.

Fig. 4 shows labor conditions and productivity in the Joplin or Tri-state district. Up to 1932 the number of men employed, the man-hours, and the number of days worked declined. The spread-work plan was in operation in 1933 as the number of men had increased, man-hours remained stationary, and the number of days worked declined still further. The increased productivity per man-hour was due perhaps largely to the

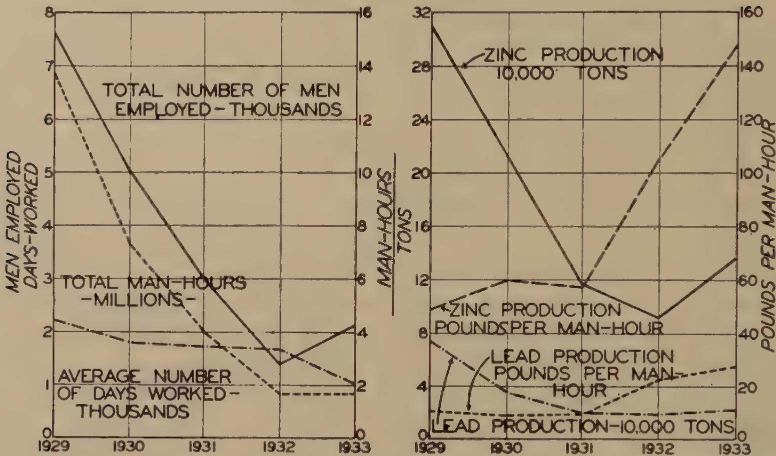

FIG. 4.—LABOR CONDITIONS AND PRODUCTIVITY IN THE JOPLIN, OR TRI-STATE, DISTRICT.

The largest zinc-producing district in the United States, ranging from 30 to 45 per cent of the country's total production. Number of men employed, man-hours and average number of days of work per year have all shown a drastic decline.

Lead is in the nature of a byproduct although it ranges from 5 to 13 per cent of the country's total. There is no byproduct value from gold or silver.

Combined tonnage for zinc and lead is the measure of labor productivity.

The graphs on these charts are for direct mine labor only and do not include milling. Source of information: U.S. Bureau of Mines.

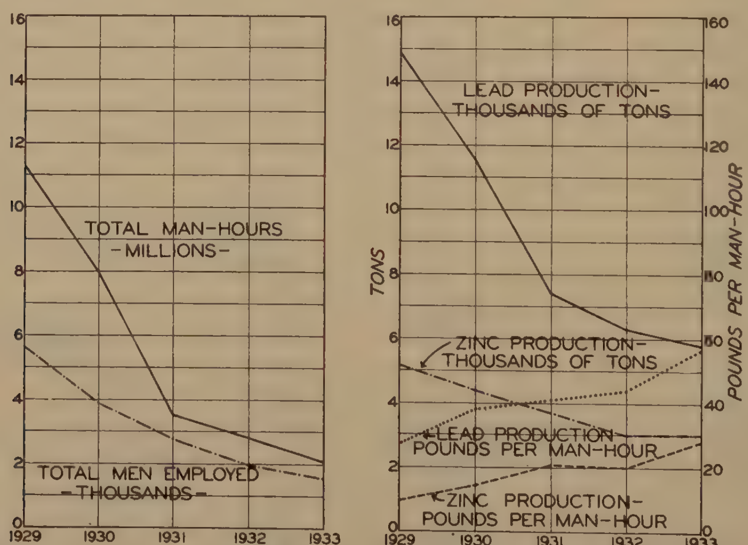

FIG. 5.—LABOR CONDITIONS AND PRODUCTIVITY IN UTAH.

In 1933 five mines produced 94 per cent of state output of lead. Almost all the zinc produced came from lead-zinc ore milled, produced by four mines. Practically no straight zinc ore has been mined since 1929. In 1932 lead-zinc ores of state contained an average of 0.098 oz. gold and 10.17 oz. silver.

Graphs are for direct mine labor only and do not include milling. Source of information: U.S. Bureau of Mines.

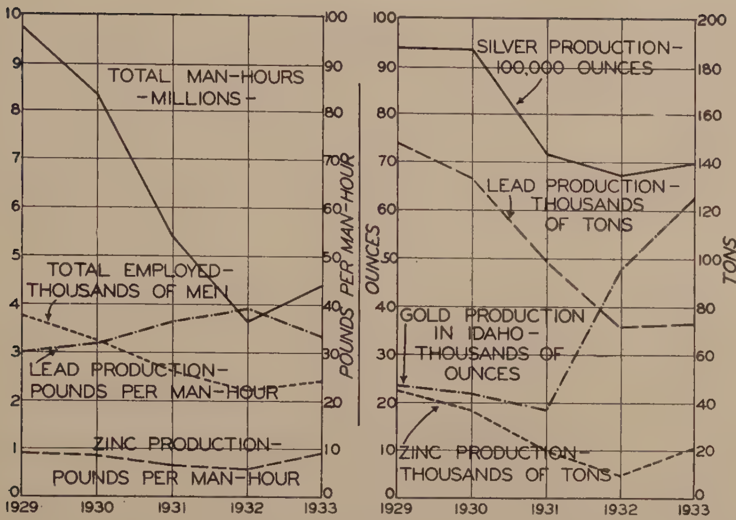

Fig. 6.—Labor conditions and productivity in the Coeur d'Alene district, Shoshone County, Idaho.

Idaho ranks second in United States (Missouri first) in production of lead. Coeur d'Alene district accounts for practically entire production of lead, zinc, copper and silver. Only a few hundred ounces of gold are produced annually in this area.

In mining complex ores containing lead, zinc, copper and silver, prices of base metals alone cannot be used in estimating profits of a mine until credit has been given for silver. The new price for silver should act as a stimulus for mining generally in the Coeur d'Alene region.

Graphs on these charts are for direct mine labor only and do not include milling. Source of information: U.S. Bureau of Mines.

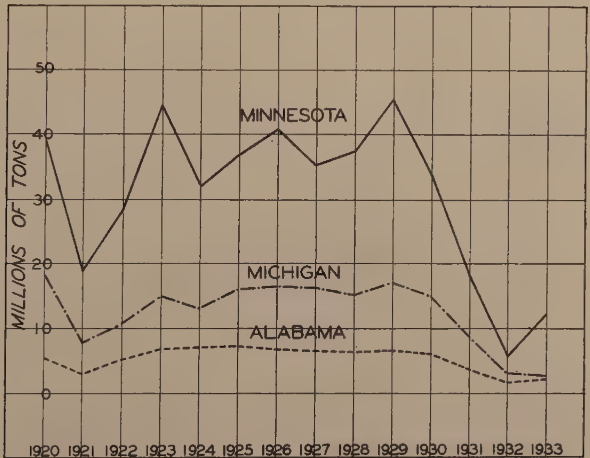

Fig. 7.—Production of iron ore in the three principal regions of the United States.

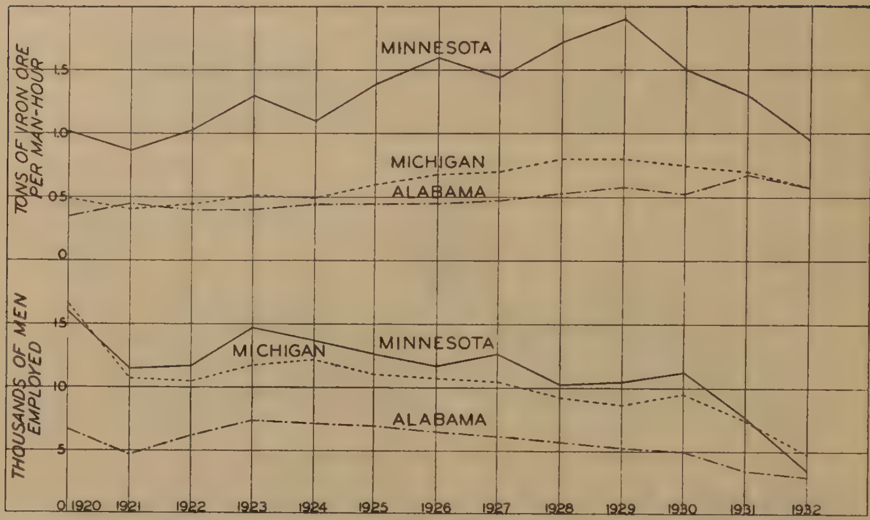

Fig. 8.—Productivity per man-hour and number of men employed in iron mining.

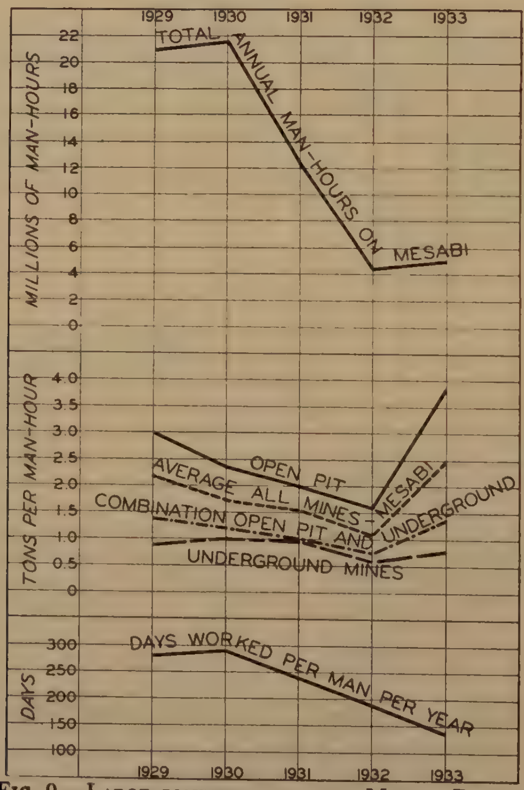

Fig. 9.—Labor conditions on the Mesabi Range.

Man-hours per year decreased sharply since 1930 with only small upturn in 1933 over 1932. More men employed in 1933, but fewer days worked per man. Tons per man-hour mounted sharply in 1933, exceeding 1929 record. Tons per man-hour for open-pit mines more than twice as high as average for combination mines, and more than three times as high as for underground mines.

mining of higher grade ore. The graph draws attention to the increased productivity per man-hour, which is the first step in determining the cause for the change. Fig. 5 shows conditions for Utah, and Fig. 6 for the Coeur d'Alene.

Fig. 7 shows the production of iron ore in the three principal regions. Fig. 8 shows productivity per man-hour. Notice how the number of

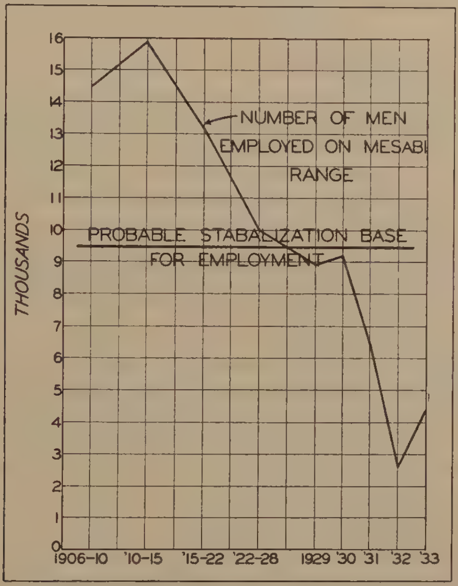

FIG. 10.—EMPLOYMENT ON MESABI RANGE.

Mechanization has steadily reduced number of men required despite mounting production. Presumably downward trend was halted in 1922, as number employed stabilized between 9000 and 10,000 for eight-year period.

Total population of Itaska and St. Louis counties, Minnesota (less pop. of Duluth), 1930 census, 129,820. Population of townships included on Mesabi Range, all virtually dependent on iron mining, 53,245. Proportion of population of Itaska and St. Louis counties included in these townships, 41 per cent. Number of men employed in iron mining on Mesabi in 1930, 9428. Number employed in 1933, 4412, or 47 per cent of those employed in 1930.

men has decreased since 1923. Fig. 9 presents a picture of labor and productivity on the Mesabi Range for 1929–1933. Fig. 10 shows how the number of men has steadily decreased despite the increased production because of the advance in mechanization. The steep drop in labor was partly halted between 1922 and 1930, which was a period of active mechanical improvement. Pending the development of other factors it is quite probable that an average of this period can be assumed as a base for future employment.

No charts are available from an individual operation. The idea, however, is commended as having possibilities that would repay any additional work or expense to the operator.

DISCUSSION

(*J. A. Church, Jr., presiding*)

J. A. Church, Jr.,* New York, N. Y.—Recently I had a curious little problem of adjusting some corporation costs from a lower wage scale to the present wage scale, in order to get a wide enough base for a forecast for the future, in which the question of man-hours came out very sharply. It was agonizing until the information was assembled. I had seven months of cost with one scale, but I needed more than a year as a triangulation basis for projecting into the future, and the only way I could adjust on a man-hour basis was to go right back to the time cards, and develop the information from the timekeeper's office.

J. B. Canada,† New York, N. Y.—I have observed an interesting sidelight on this man-hour problem in my little experience in some of the gold mines of California. I was at Angel's camp and vicinity for several years, and talked and worked with many of the miners as well as mine superintendents and managers. The thing that impressed me clearly was that, usually, where the higher wages were paid, the greater production, on a man-hour basis, there was in the mine. At the Royal gold mine, in California, where the men were paid consistently higher wages than in any of the surrounding mines, they consistently got anywhere from 3 to 5 tons more per man per day than the surrounding mines, on the basis of the same mining methods and handling. It is perhaps thought that the increase in wages would not compensate for the increase in tonnage, or that the increase in tonnage would not be sufficient for the increase in wages. It all depends largely on whether you look at it from the point of view of the miner or the mine manager, of course. But I know for a fact that in that district, usually, the higher the men are paid the more tonnage they are going to get out; consequently, costs will probably decrease.

J. A. Church, Jr.—Was there, in that case, any element of contracting?

J. B. Canada.—No, sir, that was on a straight labor basis—straight company time.

C. W. Wright,‡ Washington, D. C.—It may be of interest that the Power Commission in Washington has detailed its engineers to get the kilowatt-hours per unit of commodity output, such as iron, copper and other metals and minerals. It is starting this study so as to know what it will cost in power units to turn out a pound of copper or a ton of iron.

J. A. Church, Jr.—When you get down to units per ton of production, either of a man's time or of kilowatt-hours, you are treating costs very much as a petrologist treats a mineral under polarized light. It looks entirely different from the standpoint of charting units.

J. M. Riddell,§ Cleveland, Ohio.—I was especially interested in Mr. Holderer's chart relative to the Mesabi Range. It showed the excessive peak of 1914 and 1915, at which time stripping operations and labor requirements were at a maximum. As the stripping operations gradually diminished, the number of mining employees lessened, with the inevitable result of excess labor in the district. The situation was very uniquely coped with by those in political power to levy excess ad valorem taxes, and that condition still prevails.

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* Consulting Engineer.

† Mine Superintendent, Dewey Gold Mines Trust.

‡ Chief Engineer, Mining Division, U. S. Bureau of Mines.

§ Mining Engineer, The Corrigan, McKinney Steel Co.

Ventilation of Small Metal Mines and Prospect Openings

By Oscar A. Glaeser,* Member A.I.M.E.

(San Francisco Meeting, October, 1935)

The purpose of this paper is to present specific applications of the fundamental principles of mine ventilation and the means that are at our disposal to accomplish the task of maintaining a healthful atmosphere in small mines and prospect openings.

Need for Ventilation

The range between the maximum oxygen content of normal air and the minimum safe content is extremely narrow, approximately 5 per cent. Furthermore, normal air must not be contaminated with irrespirable gases beyond definite limits, and mineral dusts are equally dangerous, although health impairment is slower. Tables 1 and 2 present the more common respiratory hazards encountered in mines. Adequate ventilation is the most effective means available to reduce or eliminate these hazards.

Mechanical ventilation has reached varying stages of perfection in coal mines and in many large metal mines. The small mines and prospects rarely have a definite plan of ventilation. As a consequence underground workers are frequently exposed to health and accident hazards. The annual reports of fatalities in mines in the Western States record yearly many deaths due to asphyxiation and deaths caused by men falling down shafts and raises after being overcome by gas. At times mining operations are slowed up because there is no means of driving out the gas after blasting. For that reason only one shift is worked, or men wait from one to several hours for the "smoke" to clear out. Methane gas in varying quantities is widely distributed throughout California. It has been encountered in explosive concentrations in the coast range. It has been found on the Mother Lode and in quicksilver mines near the coast. Methane explosions have not been uncommon in underground operations and in tunneling jobs. These conditions and records indicate that there is room for improvement in mine-air conditions.

Manuscript received at the office of the Institute July 29, 1935.

* Mining and Safety Engineer, State Compensation Insurance Fund, San Francisco, Calif.

TABLE 1.—*Mine Gases*

Life-giving	Normal air: Oxygen 20.94 per cent, nitrogen 78.09, carbon dioxide 0.03, argon 0.94 (by volume).
	Oxygen (O_2): 20.94 per cent by vol. in normal air. Dangerous below 15 per cent (oxygen deficiency).
Nonpoisonous. Asphyxia due to oxygen deficiency.	Nitrogen (N): Inert. Accumulations above normal due to removal of oxygen from normal air. Also due to nitrogen gas feeders from rock strata.
	Methane (CH_4): Explosive (expl. range 5 to 15 per cent by vol.) Natural gas of wide geographic distribution in California.
Nonpoisonous. Asphyxia due to oxygen deficiency. Stimulates respiration.	Carbon dioxide (CO_2): A product of decomposition and combustion of organic and carbonaceous compounds. A product of blasting and burning explosives. Also a natural gas in some rock strata.
Highly poisonous even in low concentration.	Carbon monoxide (CO): A product of incomplete combustion as in mine fires, gas and dust explosions, blasting or burning of explosives.
	Hydrogen sulfide (H_2S): A product of decomposition of sulfur compounds and sulfides. A product of blasting and burning explosives. A natural gas occurring as gas feeders and carried by mineral waters.
Highly poisonous and corrosive to tissue of respiratory organs.	Sulfur dioxide (SO_2): A product of sulfide dust explosions. and of fires in sulfide mines. Also a product of blasting and burning of explosives.
	Oxides of nitrogen (NO, NO_2, N_2O_3): A product of blasting and burning explosives.

TABLE 2.—*Mine Dusts*

All mineral dusts.	All kinds of mineral dust will cause respiratory diseases, upon sufficient exposure; generally known as pneumonoconiosis. Death usually follows as the direct result of pneumonia, tuberculosis or silicosis.
Quartz or silica dust.	Upon sufficient exposure results in silicosis and death.
Dust of lead and lead compounds; zinc and arsenic compounds.	Carbonates and oxides of lead upon sufficient exposure result in lead poisoning. Ores containing zinc and arsenic in its natural forms are also poisonous, the degree of poisoning depending upon the concentrations of mineral matter and length of exposure.
Dust of ores of mercury	Mercurial poisoning or salivation. The rock particles of the dust may cause pneumonoconiosis. The results may be fatal, depending upon the severity of exposure.
Metallic mercury.	Metallic mercury is absorbed through the skin or received internally through contact, as between the hands and the mouth. Mercury vaporizes at low temperatures and may be breathed. The result is salivation and mercurial poisoning. Death depends upon severity of exposure.

When we investigate the consumption of oxygen by human beings engaged in hard physical labor and the character and quantities of gases produced by explosives, and add to that the effect of other agencies on the quality of the mine air, we are bound to conclude that an adequate air circulation in all mine workings is necessary. A man at hard labor consumes approximately 4.5 cu. ft. of oxygen per hour; the equivalent of 22.5 cu. ft. of fresh air. The air exhaled from the lungs may contain as much as 6.6 per cent carbon dioxide and only 14.3 per cent oxygen, and obviously is unfit for further human use.

Gases generated when blasting vary considerably in character and quantity, depending upon the efficiency of the shots, the amount of moisture in the drill holes, and the material being blasted. Modern explosive mixtures are calculated to produce a minimum of harmful gases. Perhaps the largest quantity of harmful gases is generated when an explosive burns. Analyses made by the U.S. Bureau of Mines of gases from blasting in sulfides showed the presence of carbon monoxide in concentrations as high as 0.78 per cent[7]. In the light of these data it is not difficult to understand why men are overcome by the so-called "powder smoke" in mine headings where there is no provision for adequate ventilation.

TABLE 3.—*Symptoms of Carbon Monoxide Poisoning*[a]

PERCENTAGE BY VOLUME	EFFECT
0.02	Will produce slight symptoms in several hours exposure.
0.04	Headache and discomfort upon 2 to 3 hr. exposure.
0.12	With moderate exercise, slight palpitations of heart in 30 min.; tendency to stagger in 1.5 hr.; confusion of mind, headache and nausea in 2 hr.
0.20 to 0.25	Produce unconsciousness in 30 min.
Higher concentrations:	effects may be so sudden that there is no warning before collapse.

[a] Advanced Mine Rescue Training, Part I. U. S. Bur. Mines *Miners Circular* 33.

The mine-gas hazard is further increased by the gasoline engine. It is against the law to use fuel-burning engines underground, but the law is not heeded. Experience has shown that the gas engine does not necessarily have to be in the mine. Frequently it is set up in a completely enclosed snowshed. The engine exhaust may be in close proximity to the mine entrance or near the compressor intake. Such conditions have on many occasions produced definite gas hazards underground. A removal of the cause—not ventilation—was, of course, the solution in such cases.

Operating efficiency is increased by the dissipation of heat and high relative humidities; also by the rapid removal of gases from blasting. I quote from the U.S. Bureau of Mines *Information Circular* 6734, by

[7] References are at the end of the paper.

Mr. Harrington, Chief of the Health and Safety Division: "Mines with high temperatures, above 75° F., and high humidity, above 85 per cent, are likely to lose from 25 per cent to as much as 75 per cent of the efficiency of workers." From *Information Circular* 6136 of the Bureau of Mines, I quote: "The principal advantage of mechanical ventilation to the operator has been the increased efficiency of the miners and in the saving of time to the men through being able to return to the working place in a shorter time after each blast."

For years warnings have been discernible on our horizon that dust in the mineral industry is a problem that will have to be dealt with very definitely or that the consequences will settle heavily upon someone's shoulders. Africa, Australia and Canada have had to face it. That this problem is now catching up with us is indicated by the claims for industrial disease (silicosis) now before our courts. Rock and mineral dust of all kinds should be eliminated as far as possible at the source. But all dust cannot be removed at its source, only adequate ventilation will remove it.

The question may be raised: what is adequate ventilation? This question can be answered definitely only in the light of known conditions and the need for emergency requirements. In mines where irrespirable and explosive gases occur it would be advisable to follow the laws of many states; namely, that the volume of air circulated at the working face should be great enough to so dilute these gases that they will be harmless. Most coal-mine states require a minimum of 100 cu. ft. of fresh air per man per minute.

Through necessity coal mines have used multiple entry systems and the entire underground scheme of coal extraction is so designed as to insure air movement at the working face. In metal mines where such a system is prohibitive other methods have been resorted to or no provisions have been made for air circulation. Quoting again from the U.S. Bureau of Mines *Information Circular* 6734: "Workers in many metal mines are much less healthy than workers in coal mines, due largely to the superior ventilation of collieries." Every effort should be made to improve the air circulation in metal mines, particularly in the dead ends, in raises, winzes and in stopes.

Natural Ventilation of Mines

Natural ventilation is relied upon a great deal. In many mines it is the only means whereby outside air enters the underground workings. When mines have more that one opening to the surface, particularly if these openings are at different elevations, there is generally some movement of air. The reversal of natural air movement in a mine, as between winter and summer or even night and day, is due to the fact that the

temperature of the mine air lies between the maximum and minimum outside air temperature. The volume of air that will flow in a given mine is dependent upon the difference in temperature as between the inside and outside air and the friction set up by the underground surfaces against the flow of air. Thus, with a 6 by 7-ft. adit 800 ft. long, and a raise of the same dimensions at the face, 150 ft. to the surface, with a mine-air temperature of 60° F. and an outside temperature of 90° F., the air may be expected to move at a velocity of 230 ft. per minute, as follows[1]:

$$\text{Unit pressure} = \frac{1.3273B(T - t)D}{(460 + T)(460 + t)} = 0.625 \text{ lb. per sq. ft.}$$

when B = barometric pressure (assumed to be 30 in.),
$T = 90°$,
$t = 60°$,
D = height of air column (150 ft.)

$$V = \sqrt{\frac{pa}{ks}} = \sqrt{\frac{0.625 \times 42}{0.00000002 \times (950 \times 26)}} = 230 \text{ ft. per min.}$$

when V = velocity per minute,
p = unit pressure = 0.625 lb. per sq. ft.,
a = area of airway 6 by 7 = 42 sq. ft.,
k = coefficient of friction = 0.00000002,
s = total rubbing surface of airway = (950 × 26).

If the outside temperature drops at night to 70° F., the air velocity will be reduced to 135 ft. per minute.

Using this natural velocity of air in the mine, shown diagrammatically in Fig. 1, what can be done to ventilate the workings? In controlling the directional flow of air we have increased its total traverse from 950 to approximately 1350 ft. This increase has added considerably to the frictional resistance, thus reducing the velocity from 230 to 194 ft. per min. during the hottest part of the day and from 135 to 114 ft. per min. at night. Other stopes could be added. The air could even be sent to a still lower level, but with each new addition the resistance rises and the velocity drops.

Natural air circulation is the result of a very small pressure differential. In Fig. 1 the maximum air motion is due to only 0.626 lb. pressure per square foot whereas the lower velocity is due to a pressure of 0.216 lb. pressure per square foot. To use this air movement for maximum benefit the limitations of the motive force must be respected. In other words, to attempt to pass an otherwise large volume of natural air through very restricted openings means that the resistance of the restricted passageway to the flow of air will act to diminish the total

volume. For example: the air velocity of 230 ft. per min. passing through 950 ft. of adit and raise of 6 by 7 ft. dimension would be reduced to 212 ft. per min. were the area restricted to 5 by 6 ft. A velocity reduction of 18 ft. may not seem much but if the volume is calculated the loss becomes

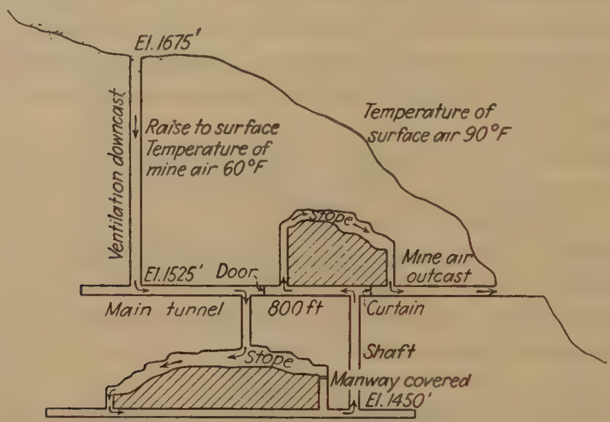


FIG. 1.—SCHEME OF VENTILATION BY NATURAL AIR COURSING.

enormous; in this case 3300 cu. ft. per min., or about 35 per cent. It is essential therefore that maximum openings be maintained for maximum benefit.

A good practical example of how a natural air current can be made to serve a definite need is shown diagrammatically in Fig. 2. A drift mine

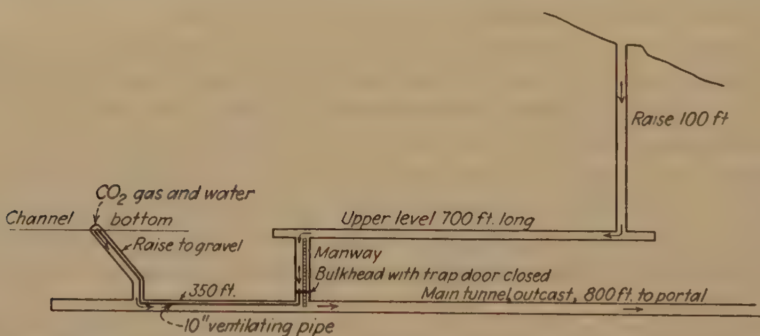


FIG. 2.—METHOD OF VENTILATING A DISTANT HEADING WITH NATURAL AIR CURRENT.

encountered carbon dioxide (CO_2) gas in a raise that tapped the bottom of a gravel channel. There was also a considerable flow of water, but even the water cascading down the irregular floor of the raise could not set enough air in motion to keep the raise clear. A carbide lamp would be extinguished near the bottom of the raise an hour after the compressor was stopped. A fair circulation of air was set up through the pipe installation shown in Fig. 2 and the problem was solved by utilizing the natural

air current. The natural flow of air was tremendously reduced, of course, but remained sufficient to keep the working face clear.

Mechanical Ventilation

Broadly speaking mechanical ventilation is air movement due to the application of mechanical power. Such a definition would draw no distinction as between the restricted movement of air through ventilation pipe and the coursing of the air through the mine workings. To differentiate between the two the former will be considered under Auxiliary Ventilation.

Relatively few metal mines, and then only the larger ones, employ mechanical ventilation. Some of these have developed elaborate ventilation systems for removing gases from blasting, removing dust, decreasing temperatures and high humidities and producing a cooling effect through air motion. Further elaboration has been deemed advisable in air control for emergency purposes such as mine fires. Other mines could elaborate on their systems or install mechanical ventilation with considerable profit in increased efficiency, lower labor turnover, fewer accidents and better health among the mine workers.

The mechanical ventilation of mines by coursing the air through the entire mine requires a fan, a fresh-air intake and a mine-air outlet. The fan in motion sets up the pressure differential which in natural air movement is developed through the difference of the inside and outside air temperatures. Directing and controlling the movement of air in the mine becomes a matter of doors, stoppings and possibly regulators. Such installations should be as few as possible, since generally they interfere with operations. On the other hand, enough doors should be used, particularly in mines containing much timber. The writer knows of one door in a large western mine that was worth its weight in gold. It had not been used for several years, but one night when a fire threatened the entire upper portion of the mine, that door, after it was closed by a courageous helmet crew in a blistering heat, stopped a fast-moving fire by shutting off the draft.

The question of whether the fan shall exhaust the mine air or blow the fresh air into the mine will have to be determined. This is a problem to be seriously considered in the light of existing conditions, future development and possible emergencies. An exhaust system lowers the mine-air pressure to below atmospheric pressure. A blowing system raises the pressure above atmospheric. An exhaust-fan system should be so designed that fan controls will be in fresh air regardless of where a fire may break out. The fresh-air intake, whether the system is blowing or exhausting, should have no inflammable structures or materials near it. All underbrush and forest growth should be cleared away for a distance that will insure against smoke in dangerous concentrations being drawn

into the mine. The intake itself should be as nearly fireproof as it is possible to make it. The main hoisting shaft or haulage tunnel should, if at all possible, be kept free of incumbrances such as doors. This generally means that another outlet is connected to the fan. If at all possible the main fan should be on the surface and in a fireproof structure. An underground fan may not be accessible in case of fire.

The size of the fan depends entirely on the ventilation load. Since primary fans are considered permanent installations, due consideration should be given to future mine development before the fan size is decided.

Fan power consumption will be a new item of expense on the cost sheet. There is a direct relation between the volume of air delivered, the mine resistance that must be overcome, and the power applied to the fan. For example:

$$\text{Horse power} = \frac{\text{unit pressure} \times \text{velocity} \times \text{area of airway}}{33{,}000}$$

Doubling the velocity quadruples unit pressure. So that in the formula above if the airway remains unchanged but the velocity is doubled, resulting in a four times greater pressure, the horsepower must be increased eight times in order to deliver the load imposed. On the other hand, if the volume remains unchanged but the velocity is cut in half by virtue of a greater airway area, the horsepower required will be only approximately one-sixth of that formerly required. The reason it is not the inverse of the power increase when doubling the velocity is that the velocity was reduced one-half by enlarging the airway. This, naturally, increased the rubbing surface, thus consuming some additional power. Hence the drop in the power is only five-sixths instead of seven-eighths. There is considerable difference between the power bills of a 50-hp. and a 300-hp. motor. Over a period of time considerable airway expansion may be justified in the saving that will result. Restrictions in airways that seriously impede the flow of air should be removed.

Ventilation Supervision

The operation of a large mine fan costs money. Ventilation therefore deserves its share of supervision. Even in the smaller mines employing mechanical ventilation some individual should be charged with the scheme of ventilation. Only a mine official in charge of the entire mine can effectively supervise a ventilation system. In large mines an engineer should be assigned to the work. Air currents should be indicated on the mine maps. The maps should also show all doors and other appurtenances.

Auxiliary Ventilation and Equipment

Air currents follow lines of least resistance and therefore take the shortest route available. Dead ends of drifts and stopes without a second outlet are passed by. Under certain conditions an induced air current is set up in drifts or tunnels. This is particularly noticeable when there is a great difference in temperature or a considerable flow of water. However, such induced circulation does not extend inward great distances.

In some mines auxiliary fan equipment is used to supply the mine with fresh air when only one opening exists. At other mines the natural air current or mechanical ventilation is used for general air circulation and where necessary auxiliary equipment is installed to ventilate dead ends. Auxiliary ventilation is the forced movement of air through pipe or tubing. The air may be directed to one working face or, by means of

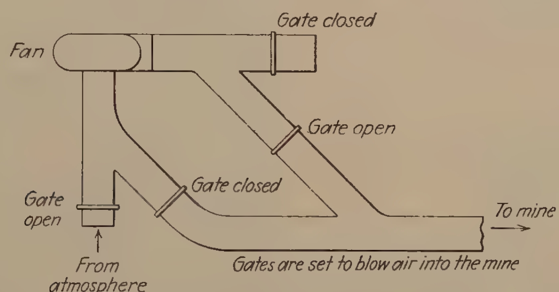


Fig. 3.—Auxiliary fan connected to reversible system of pipes and gates.

branches, it may be distributed to a number of faces. It may be blown in or exhausted or, by means of a reversible system of pipes and gates, may be either drawn out or blown in at will. See Fig. 3. The ventilation pipe is carried as close as practical to the working face without being exposed to possible damage from blasting. A common practice in some mines is to carry the pipe to within 100 or 150 ft. of the face and from there on use flexible tubing suspended by wire or hung on the water or air line. The flexible tubing is removed at blasting time. I favor this method because it delivers the air where it is most needed and because it has a definite cooling effect on the workmen at the face. This effect is completely lost with an exhaust system. When the fan is used as an exhauster immediately after blasting, the best results are obtained by allowing the compressed air to blow at the face. The compressed air tends to force the gases back to the end of the fan pipe, where they are picked up and drawn out. The flow of compressed air should be regulated to approximately coincide with the volume of air drawn into the fan pipe.

Many mines depend entirely upon compressed air to supply the headings with air and clear the gases from the face after blasting, and, of course, miners can hardly avoid breathing it while running their machines.

Nevertheless it is not the best air by any means, as there is always a tendency toward contamination from the oil in the compressor. Recently an accident occurred, resulting in a fatality, that will serve to illustrate this point. One of the valves in a compressor stuck, causing the compressor to run hot. The heat and pressure apparently broke down the lubricating oil, generating, among other gases, carbon monoxide. Some of the air from this compressor was used by men working under sandblaster's hoods. Apparently these men did not notice anything wrong until after they had inhaled considerable carbon monoxide, which caused one man's death. The use of compressed air as an only source of air for breathing purposes should be discouraged.

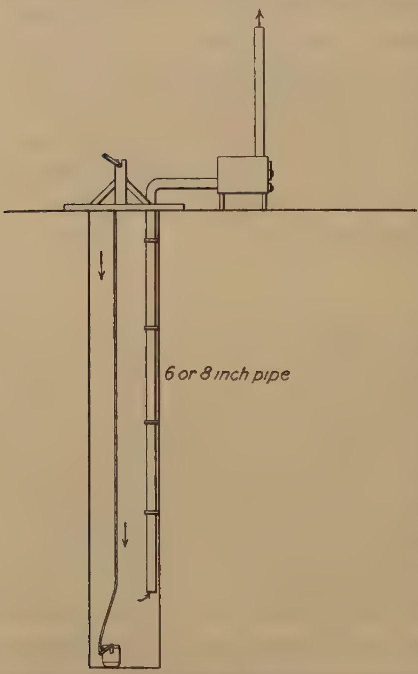


FIG. 4.—VENTILATING SHALLOW SHAFT WITH STOVE AND PIPE.

Many types of auxiliary blowers and other devices designed to impart movement to air are used. One of the most ancient methods of ventilation is perhaps with the aid of fire, which is still used today, particularly in sinking shallow shafts (Fig. 4). It is slow and crude but in isolated places it has served a very definite and useful purpose. It is inefficient because a brisk fire must be maintained to obtain any kind of result. Thus it needs constant attention and fuel consumption is high. It is not recommended as a method of ventilation, but is mentioned as a matter of historic interest.

The water blast is also still in use and several ways of using it have been developed with varying degrees of success. The greater the dimensions of volume of water, size of pipe, and length of fall, the greater the volume of air delivered. A good volume of air can be forced through several hundred feet of 8-in. ventilation pipe by a full flow of water passing through a 2-in. funnel and dropping 40 to 50 ft. in a vertical 10-in. box or pipe. The free falling water type is generally used to blow air into the mine, although in its reversed form it may also be used to exhaust the air. Applications of both methods are shown in Figs. 5 and 6. The injector principle, using a jet of water, has also been used in at least one place, and with surprising results. Air was drawn through 3000 ft. of 8-in. galvanized iron pipe with a velocity that would extinguish a candle at the pipe intake. Fig. 7 is a diagrammatic sketch of this type. At one drift

mine, the mine operator utilized the drainage water from a level above, as shown in Fig. 8.

An injector blower using compressed air was first used on the Witwatersrand and there is known as the Modder blower. It was further

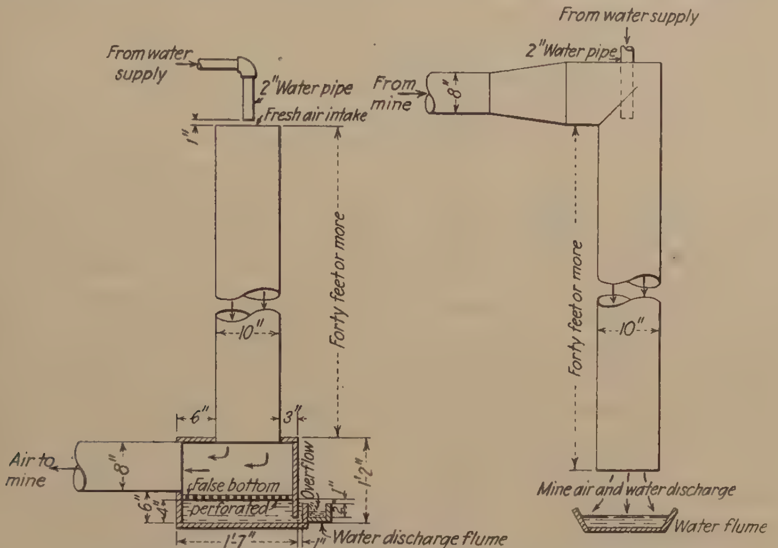


FIG. 5.—AIR BLAST ARRANGED TO BLOW AIR INTO MINE. FIG. 6.—AIR BLAST ARRANGED TO EXHAUST AIR FROM MINE.
FIGS. 5 AND 6.—DETAILS OF WATER BLAST.

developed by the writer while Safety and Ventilation Engineer for the United Verde Copper Co. and is now used quite extensively throughout the Western States. Fig. 9 gives details of the blower, together with dimensions for several pipe diameters. The advantages of the injector

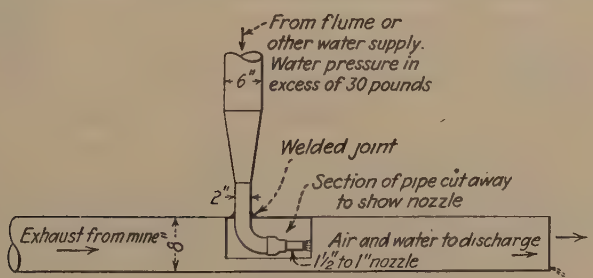


FIG. 7.—WATER INJECTOR.

blower are its low cost, portability, compactness and the volume of air it will deliver. Its disadvantage lies in the fact that it uses compressed air. A number of mine operators in California are enthusiastic about this type of blower.

The compressed-air driven, rotating blowers consist of the axial flow or modified propeller type and the centrifugal type. Both are high-speed

machines of 3400 to 3600 r.p.m. The centrifugal type develops the higher pressure and is therefore more practical for long-distance air delivery.

Electrically driven blowers can be obtained in many sizes and designs. The tendency in recent years has been to the direct connected high-speed

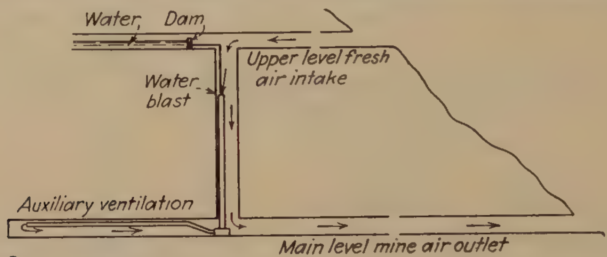
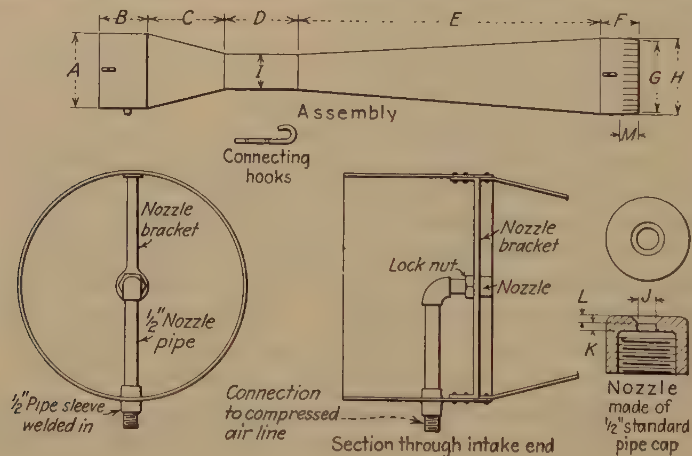


FIG. 8.—UTILIZING DRAINAGE WATER TO VENTILATE MINE HEADING.

machine having non-overload characteristics. The high-speed blowers have passed through the experimental stage and are now thoroughly reliable. The motors are rugged and designed to carry the load. Their high speed has made possible the smaller and lighter machine, generally



Diameter	A	B	C	D	E	F	G	H	I	J	K	L	M
12 inch	12	8	12	12	48	6	11¾	12	5½	¼	3/32	3/32	3
8 inch	8	5	6	6	30	4	7⅞	8	3⅜	⅛	3/32	3/32	2
6 inch	6	3	3	3	18	2	5⅞	6	2	⅛	3/32	3/32	1

Dimensions of injector blower

Nozzle diameter, inches	⅛	3/16	¼	⅜
Compressed air, cu ft per thousand, free air	25	44	54	100
Volume delivered through 200 ft, 10-in. tube		2100	2700	3200

Compressed air consumption and volume of air delivered for various sizes of nozzles

FIG. 9.—INJECTOR BLOWER.

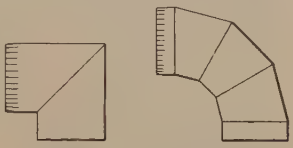
known as a portable blower. Yet their capacity both in volume and pressure developed is greater than the much more cumbersome slow-speed units. The high-speed blowers are also made for belt drives.

Positive pressure blowers are used by the mining industry for ventilation purposes. They become particularly useful for long-distance jobs where air must be delivered through 2000 ft. or more of ventilation pipe.

They begin to serve effectively when centrifugal blowers fail against the resistance of a long pipe line.

The question of the size of blower to purchase should be decided in favor of a somewhat larger rather than a smaller unit. The power consumption of any of these fans except the positive pressure blowers is small—less than 5 hp., although some fans come equipped with a 5-hp. motor. It should be remembered that the smaller the fan, the more limited its use becomes. In mining, equipment of broad utility is favored, as conditions vary from month to month. Thus the author would be inclined to favor a high-speed blower with a volumetric capacity of 2000 to 3000 cu. ft. per minute against a resistance of 5 to 6-in. water gage. A fan of this size can be used to ventilate several faces through branch pipes where the distances are not too great or where the total distance does not exceed 1500 feet.

Air ducts are as important as the blowers. Galvanized slip-joint pipe is most commonly used. Common diameters are 6, 8, 10, 12 and, less frequently, 16 in. Weight of the sheet iron varies from 10 to 20 gage. For long tunnel jobs, when positive pressure blowers are used, the pipe is generally flanged and is bolted together. In Canada, pipe with one end belled out and the other straight has become popular. The wedge-shaped space is filled with okum or other packing. Its advantage lies in the fact that there are no constrictions, as in crimped pipe. The inside of the pipe offers a continuous smooth surface to the flow of air. Easy curves and bends are made without difficulty. All pipe joints should be wrapped for airtightness. A cheap muslin dipped in P. & B. paint, or each layer painted with the paint, makes a more lasting joint than when cement is used for the purpose. The cloth should be held in place with at least two bands of wire. Sharp corners should not be used. Instead bends should be placed in the line where corners cannot be prevented (Fig. 10).



a. *b.*

a. Sharp 90° elbow, poor practice

b. 90° segment bend, good practice

FIG. 10.—METHOD OF TURNING VENTILATING LINE AT CORNERS.

Pipe diameters should be large rather than small. For example, let it be assumed that a fan is delivering 1000 cu. ft. of air per minute through an 8-in. ventilation pipe. If a 6-in. pipe had been used under similar conditions, the volume delivered by the same fan would have been only about 475 cu. ft. per minute, but using a 10-in. pipe the volume of air delivered would have been stepped up to 1750 cu. ft. per minute. The initial cost will be greater, of course, for the 10-in. pipe, but ventilation pipe lasts for years and its value lies not in its size but in the service it renders. Generally speaking, the tendency is toward pipe of too small a diameter.

In small mines, flexible tubing is not as commonly used as sheet-metal pipe; perhaps because a sheet-metal shop can be found in the nearest town whereas flexible tubing is not so readily available. Flexible tubing is of practical value to companies doing much exploration work and moving from mine to mine. The tubing can be rolled up in compact bundles and transported without difficulty. A comparative study of metal pipe and flexible tubing shows each to have definite advantages, with the cost item in favor of flexible tubing[6].

Dust Control

It is not the purpose of this paper to delve into the technology of this important subject. The rock and ores of practically all metal mines contain free silica in varying quantities and therefore produce rock dust

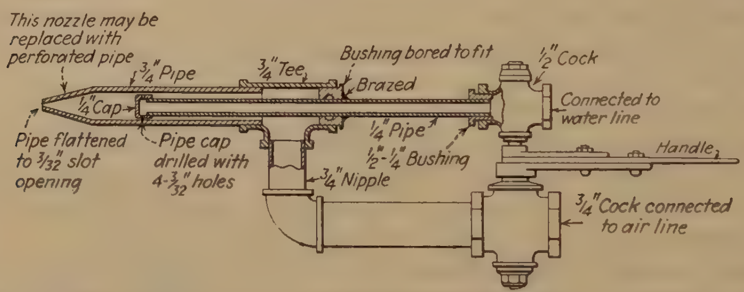


Fig 11.—Combination air and water spray for dust and powder gases

containing free silica. Various mining operations such as drilling, blasting, mucking, dumping, pulling chutes and transportation of rock and ore have a definite tendency to make dust. The problem we are confronted with is how to get rid of the dust. The most effective way of keeping the dust out of the air is to deal with it at every point where it is given an opportunity to rise. Dry drilling should not be tolerated. Muck piles should be thoroughly wet down. Water sprays at dusty chutes will help to allay the dust at such points. Water sprays have also been used to good advantage near headings. They are turned on immediately after blasting.

Blasting creates more dust than any or all mining operations put together. The spray will help to allay the dust, but good ventilation is required to move it out. It takes time even with a good air current to remove this finely pulverized dust from the mine. At one large and particularly well ventilated metal mine, it was determined by dust count that 45 minutes is required to get the dust content back to normal after blasting. In a poorly ventilated mine, or one in which no ventilation is provided for dead ends, the dust may remain suspended for hours.

Good ventilation, ample water, no blasting or a minimum of blasting while men are on shift, and a short rest period between shifts are the methods at our command to combat the dust hazard in our mines.

Mine Fires

A discussion of mine ventilation is not complete without some attention to mine fires. In practically every instance where mine fires have taken a toll in life, ventilation control designed for emergencies might have averted the tragedy. We need not go beyond the borders of our own state for harrowing experiences. The State Safety Orders require that no inflammable structures shall be built or inflammable materials stored within 100 ft. of any mine entrance. There is adequate reason for such a Safety Order, although in general it is little understood and often disregarded.

The gasoline engine is a constant menace as a fire hazard. It is frequently set up in a wooden structure very close to the main mine entrance or in a wooden snowshed that covers the mine entrance. At times the snowshed is enlarged and contains timber storage or hay for the mine mule. Sometimes the snowshed is connected with other wooden structures, such as the mill, a blacksmith shop, change house, bunk and boarding houses. In every case a serious outside fire hazard has been created that imperils the lives of underground workers in case of fire. When these hazards are pointed out it is frequently argued that the men could not be trapped and that there never has been a fire on the premises. The fact that only a few years ago five men died in a mine within sight of daylight, which they could not reach because of the drift of the deadly carbon monoxide gas from an outside fire, has long since been forgotten. As for fire experiences, there are many each year. In one very recent year the following fires in the State of California came to the author's attention: (1) a change house and snowshed, (2) several bunk houses, (3) one large mill, (4) two small mills, (5) a snowshed, timbershed, and a winter's supply of hay and straw.

The more serious underground fire hazards in the smaller mines are (1) rubbish piles in or near powder magazines, made up of broken powder boxes, paraffined paper and sawdust; (2) electric wiring. Electric wires are supposed to be sheathed in conduit or else armored cable should be used, yet frequently wiring installed many years ago is hooked up and used after long periods of idleness. The open lamp and the acetylene torch present fire hazards that should be recognized.

To overcome some of these hazards it would be well to install a fireproof door near the mine entrance, which could be closed in case of a surface fire or to control the draft in case of a mine fire. The powder magazine should be fireproof. Certainly a fireproof door should be used.

Electric wires should be encased in conduit or armored cable should be used. A second exit should be provided.

In mechanically ventilated mines a definite system of fire control should be worked out. The fan should preferably be of the reversible type. Exits should be marked and the men acquainted with them. Heavily timbered shafts can be protected with a sprinkler system made of perforated pipe and connected to the surface water lines, at small cost.

Mine ventilation is a very broad subject. This paper has merely skimmed over those features that are most applicable to our western gold mines and other small mines and emphasized certain points that several years of inspection experience by the author indicate the need of emphasis.

The U.S. Bureau of Mines has pointed the way to health, safety, and efficiency in mining through its field work and its many practical publications on mine ventilation and allied subjects. Only a few of its excellent publications are mentioned here. A brief bibliography follows.

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DISCUSSION

(*Wilbur H. Grant presiding*)

S. H. Ash,* Berkeley, Calif. (written discussion†).—I am going to call your attention to some of the problems on the Pacific Coast, especially at the smaller mines, that seem to bear on ventilation.

Ventilation is certainly a very broad subject. One of the reasons we recognize for its application is to reduce the dust about which we are talking, also to remove the poisonous and noxious gases from mines, and also to lower the temperatures and in some cases the humidity. Some of our mines on the Pacific Coast have very high temperatures. In certain tunneling operations we have found them pumping air as high as 150° F. dry bulb to the tunnel workers. That was usually done with a blower system. Many of the drifts and small mines and tunnels in this region are operated with only one opening and we have to depend upon the blower ventilating system. The reason for the 150° air temperature was that they utilize more or less large lengths of metal pipes out in the hot sun, with temperatures of 120° and more in the sun. Types of ventilators that raise the temperature internally some 25° or 30° are in use, so that within a reasonable distance, at least half a mile and at times even more, they were actually pumping air to those workmen with a temperature of 150° dry bulb.

You can imagine what takes place in a large stope or in a large tunnel section when, even at that high velocity, air is delivered to the workers at such temperatures. Where you are drilling wet and have a wet working place, almost immediately the humidity builds up to near saturation. The air current moves rather slowly and almost if not unbearable temperatures result.

The remedy in such instances is often to simply reverse the direction of air flow; in other words, to operate on an exhaust system rather than a blower system. Unfortunately, in tunnel and metal-mine practice, this matter of exhausting or reverse blowing is not also used to eliminate the dust. I have seen frequently, after blasting, especially in large stopes and tunnels, the air move out laden with dust at the time of the blast. The dust goes outby in the air current and as a result thousands of feet of tunnel are full of air that contains very fine and dangerous dust. The exhaust system of ventilation would remove this dust-laden air. All you have to do is to look at the fan at time of exhausting after blasting and see the dust coming out through the tunnel; an exhaust system remedies this.

With rock temperatures of around 80 to 85° F. by sucking the air in, or in exhaust systems, the temperature was reduced in a short time to rock temperatures. When you have such a relatively low temperature at the face, by providing auxiliary blowers you give the air motion, which after all is the essential thing in such instances.

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* District Engineer, Safety Division, U. S. Bureau of Mines.

† Published by permission of the Director, U. S. Bureau of Mines.

A little fire happened a few years ago which was not such a little fire in its effects, as it killed five men in one of our mines in the Sierras. That accident would not have happened if the ventilation had been controlled. The men could actually look outside; this fire occurred on the surface, we do not know how, burning the surface buildings. The loss of five lives could have been prevented if there had been a door in that part of the workings. The drift was in solid rock and if there had been such a door the men could have closed it and controlled the ventilation, at least insofar as excluding the fire gases.

What seems to be the big problem in most of our metal mines is the lack of control of ventilation. The things mentioned happen more often than is realized. Just last week I stopped to investigate an accident in which two men were killed and four other men knocked out, and it is a wonder that some of the four did not die as a result of fumes from blasting. In some of our mines we use considerable explosives. They may all be good when obtained but occasionally some of them may not be in proper condition at the time they are used on account of the extremes of temperature or humidity or length of time to which they may have been subjected in storage or handling.

I have seen explosives lying out in the hot sun in temperatures sometimes of 130° or 140° F. Later on this "powder" is taken underground and used under conditions which it was never intended to withstand. I have seen holes loaded full to the collar and after blasting many of the paper wrappers were stuck against the roof. There is very likely to be much poisonous gas in that atmosphere. Sooner or later (and it happens far too often) our workmen are killed by such fumes; this happened only last week in a California mine.

My attention has been called recently to three different mines where men have been overcome by fumes after blasting. This is one of the very important reasons for efficient ventilation, not only to get rid of the poisonous and noxious fumes but to get rid of the dust that is set up under these conditions.

One of the most important things that we have to contend with in this state, particularly around our small mines, is the matter of fire prevention. This is due in some manner or other to the lack of controlled ventilation. We also have instances of use of gasoline-burning equipment underground although it is against the state law. Unquestionably one of the reasons for the frequency of occurrence of accidents in many small mines is that small partnership mines do not come under the safety acts. The inspectors have no jurisdiction over them and one can readily realize that in the matter of controlled ventilation conditions are not likely to be what they ought to be.

All of these things, gentlemen, are leading up in a good many ways to dust and other occupational diseases caused by breathing the dust and noxious gases that are in the mines.

Mr. Glaeser covers the subject very thoroughly.

Cooling Effect of Compressed Air When Freely Expanded

By Walter S. Weeks,* Member A.I.M.E.

(New York Meeting, February, 1937)

The process of cooling air by allowing it to expand and do work in an engine is well known, but the theory of obtaining cold air by free expansion without the aid of an engine operating with cutoff has been little discussed.

Case I

If compressed air is allowed to expand to the atmospheric pressure through an orifice in an air line that is supplied by a running compressor, no cooling results except the negligible one due to the Joule-Thompson effect, which will not be considered in this discussion. The effect of moisture will also be neglected.

Symbols

P_1 Absolute pressure in pipe line, lb. per sq. ft.
P_3 Atmospheric pressure, lb. per sq. ft.
V_1 Volume occupied by 1 lb. air in pipe.
V_3 Volume occupied by 1 lb. air at P_3.
C_v Specific heat of air at constant volume, B.t.u. per lb. per deg. F.
C_p Specific heat at constant pressure, B.t.u. per lb. per deg. F.
T_1 Absolute temperature of air in pipe, deg. F.
T_3 Absolute temperature of air outside after expansion and mixing and coming to rest.
J Mechanical equivalent of heat (778 ft. lb.)
R Gas constant for air, ft-lb. per deg. F. = 53.3.

Formulas

Consider 1 lb. of air moving out of a pipe. The air behind it does work on it to the extent of P_1V_1 ft-lb. This work is really done by the thrust of the piston of the compressor and does not come from any energy that exists in the air in the pipe. The pound of air contains

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* Professor of Mining, University of California, Berkeley, Calif.

internal energy to the amount C_vT_1. The total energy that must appear in some form outside of the pipe, expressed in heat units, is

$$C_vT_1 + \frac{P_1V_1}{J} + \text{kinetic energy of compressed air in pipe.}$$

The kinetic energy is usually so small that it will be neglected.

After the air gets outside it has expanded and done work on the atmosphere to the extent of P_3V_3 ft-lb. Its temperature is now T_3. The total energy accounted for outside is

$$C_vT_3 + \frac{P_3V_3}{J}$$

This must be equal to the original energy

$$C_vT_1 + \frac{P_1V_1}{J} = C_vT_3 + \frac{P_3V_3}{J}$$

Since for 1 lb. $PV = RT$,

$$C_vT_1 + \frac{RT_1}{J} = C_vT_3 + \frac{RT_3}{J}$$

$$R = (C_p - C_v)J$$

$$C_vT_1 + \frac{(C_p - C_v)JT_1}{J} = C_vT_3 + \frac{(C_p - C_v)JT_3}{J}$$

$$C_pT_1 = C_pT_3$$

$$T_1 = T_3$$

In the orifice the air expands adiabatically and does work on the air in front of it by making it move faster, so internal energy is transferred into kinetic energy, which appears as heat when the air comes to rest.

While the air is flowing rapidly it is cold, but its temperature cannot be measured with a stationary thermometer. The only way that the temperature of a rapidly moving gas could be measured directly with a thermometer would be to have the thermometer moving at the velocity of the gas.

Case II

If compressed air is in a closed tank and this trapped air is allowed to flow to the atmosphere, adiabatic expansion takes place in the tank and the air therein becomes progressively colder as the pressure drops. As each bit of air leaves the tank, what corresponds to the P_1V_1 term of case I is supplied, not by the thrust of the compressor piston but by the internal energy of the air in the tank at the time. Each bit of air that leaves the tank, when brought to rest in the atmosphere, returns to the temperature it had when about to leave. Each bit of air that leaves is colder than the portion before, because adiabatic expansion is taking place in the

tank. The net result is that the average temperature of all the air that flows out, when outside and at rest, is lower than its original temperature in the tank at the start of expansion.

Additional Symbols

- W Total weight of air in tank when full of compressed air, lb.
- w Weight remaining in tank after expansion to atmospheric pressure, lb.
- P_2 Absolute pressure in tank at start of expansion, lb. per sq. ft.
- T_2 Temperature of air in tank at start of expansion, deg. F. abs.
- T_0 Temperature of air in tank at end of expansion, deg. F. abs.
- V Volume of tank, cu. ft.
- V_3 Volume of air that flows out at pressure P_3 and at rest.

Formulas

The loss of internal energy from the tank is equal to the internal energy of the air that passes out plus the external work done on the atmosphere.

$$C_v(WT_2 - wT_0) = C_v(W - w)T_3 + \frac{P_3V_3}{J}$$

$$P_3V_3 = (W - w)RT_3 = (W - w)(C_p - C_v)JT_3$$

Hence:

$$C_v(WT_2 - wT_0) = C_p(W - w)T_3$$

$$\frac{C_p}{C_v} = \gamma = 1.4$$

$$T_3 = \frac{1}{\gamma}\left(\frac{WT_2 - wT_0}{W - w}\right)$$

$$T_3 = \frac{\frac{1}{\gamma}\left(\frac{P_2V}{RT_2} \times T_2 - \frac{P_3V}{RT_0} \times T_0\right)}{\frac{P_2V}{RT_2} - \frac{P_3V}{RT_0}}$$

$$T_3 = \frac{(P_2 - P_3)(T_2T_0)}{\gamma(P_2T_0 - P_3T_2)} \qquad [1]$$

The temperature T_0 is obtained from the formula for adiabatic expansion

$$\frac{T_2}{T_0} = \left(\frac{P_2}{P_3}\right)^{\frac{\gamma - 1}{\gamma}}$$

Example

Compressed air in a tank has a gauge pressure of 90 lb. per sq. in. and a temperature of 90° F. The atmospheric pressure is 14.7 lb. per sq. in.

Assuming no heat exchange with the outside through the tank walls, what will be the temperature of the air that flows to the atmosphere after mixing and coming to rest?

From the equation for adiabatic expansion the temperature T_0 in the tank after expansion will be 312° F. abs.

$$T_3 = \frac{(104.7 - 14.7)(550)(312)}{1.4(104.7 \times 312 - 14.7 \times 550)} = 450° \text{ F. abs. or } -10° \text{ F.}$$

(On account of the form of the formula, pressure may be expressed in pounds per square inch or in pounds per square foot.)

Offhand, one might infer that to refrigerate air it is necessary only to admit air to a tank, shut it off from the source, and expand at once to atmosphere. The obstacle lies in the fact that when the compressed air enters the tank its temperature rises.

Case III

Compressed air flows from a line into a tank at a certain pressure and temperature. What will be the temperature in the tank when full of air at pressure of the line?

Symbols

- W Weight of air in tank when full of compressed air.
- w Weight of air in tank at start of admission.
- P_2 Absolute pressure in tank when full of compressed air.
- P_3 Absolute pressure in tank at start of admission.
- T_1 Temperature of air in line, deg. F. abs.
- T_2 Temperature of air in tank when full of compressed air at line pressure, deg. F. abs.
- T_0 Temperature of air at atmospheric pressure in tank, deg. F. abs., before compressed air is admitted.
- W_1 Weight of air flowing in $= W - w$.

Formulas

The energy entering the tank from the line is $W_1C_pT_1$. Since no external work is done in the tank this must appear as change in internal energy in the tank.

$$W_1C_pT_1 = WC_vT_2 - wC_vT_0 = C_v\left(\frac{P_2V}{R} - \frac{P_3V}{R}\right)$$

$$W_1 = \frac{V}{R\gamma T_1}(P_2 - P_3)$$

$$W = W_1 + w = \frac{V}{R\gamma T_1}(P_2 - P_3) + \frac{P_3V}{RT_0}$$

The temperature in the tank may be obtained from the formula

$$P_2V = WRT_2$$

$$P_2V = \left[\frac{V}{R\gamma T_1}(P_2 - P_3) + \frac{P_3V}{RT_0}\right]RT_2$$

Whence

$$T_2 = \frac{\gamma P_2T_1T_0}{(P_2 - P_3)T_0 + \gamma P_3T_1} \qquad [2]$$

If $T_0 = T_1$ the formula follows:

$$T_2 = \frac{\gamma P_2T_1}{P_2 + 0.40P_3} \qquad [3]$$

Example

Compressed air at 90 lb. per sq. in. gauge is taken into a tank. The air is cooled to the atmospheric temperature 90° F. This air is then released. To what temperature will the air rise when the tank is again filled?

From example 1, the temperature in the tank T_0 will be 312° F. abs. after expansion.

$$T_2 = \frac{1.4 \times 104.7 \times 550 \times 312}{90 \times 312 + 1.4 \times 14.7 \times 550} = 638° \text{ F. abs. or } 178° \text{ F.}$$

Résumé of Formulas

Pressures may be expressed in any units. Temperatures must be absolute and either Fahrenheit or centigrade.

To determine the temperature of air exhausted to atmosphere from a closed tank:

$$T_3 = \frac{(P_2 - P_3)(T_2T_0)}{\gamma(P_2T_0 - P_3T_2)}$$

To determine temperature of air in tank after compressed air is admitted:

$$T_2 = \frac{\gamma P_2T_1T_0}{(P_2 - P_3)T_0 + \gamma P_3T_1}$$

If temperature of air in tank at start of admission is the same as that of incoming air, this formula reduces to

$$T_2 = \frac{\gamma P_2T_1}{P_2 + 0.40P_3}$$

Upon admission, the actual temperature will not rise as high as the theoretical because of heat exchange with the metal of the receiver. If the valve from the line is closed as soon as air is admitted, the pressure will drop as the air cools, and so when the air is released the rates of expansion will not be as great as is possible. The air line should not be closed until the air in the tank has cooled. A tank with a diameter of 2 ft. and height of 4 ft. will cool, after admission of air, to within 10° F. of the room temperature in about five minutes. On release, the air takes on heat from the tank as soon as it begins to expand, so release should be rapid through an insulated pipe.

In order to make use of these principles for supplying cooled air, it would be necessary to place the tank a little distance away from the place to be cooled and provide some means of removing the heat generated by the admission of air.

The cooling of the exhaust of a direct-acting water pump follows the laws that have been discussed. The air entering the cylinder does not heat, however, because the PV term that causes the heating when air flows into a tank is used in lifting the water and forcing out the remaining exhaust air. At the end of the stroke the cylinder is full of compressed air at the temperature of the line. The exhaust valve opens and the air is released to the atmosphere. This is identical with releasing from a closed tank and the resulting cooling is the same. The temperature of the air remaining in the cylinder is the same as in the tank after adiabatic expansion. On the return stroke, this cold air is forced out into the atmosphere. No means existed for getting this cold air out of the tank. It served merely to help cool the air after another admission.

The pump, then, has two advantages over the tank as a cooling mechanism: (1) it permits continual operation; and (2) the cold air at end of expansion is forced out.

DISCUSSION

(Gerald Sherman presiding)

M. Mosier,* Pittsburgh, Pa.—I should like a little more information on the effects developed by a rock drill. Recently in some tests at the U. S. Bureau of Mines research adit, with an atmospheric temperature of 25° F. and a rock temperature of 54°, we observed that the exhaust from the rock drill registered 0°.

I do not know whether or not I understood you correctly, that the exhaust from a rock drill has no cooling effect, but, nevertheless, if you stand in the line of the exhaust, you observe a decided cooling effect.

P. Bancel,† New York, N. Y.—If you expand air through an orifice, the energy to give that volume of air velocity has come from heat. The author points out that

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† Assistant Chief Engineer, Ingersoll-Rand Co, Mr. Bancel presented the paper.

the only way one could measure the effects would be to have a thermometer go along with it. In a rock drill, when the air is rushing out, momentarily the air is very cold, but after the velocity has been lost in turbulence the coldness is lost.

We build ejectors where we use a steam jet or an air jet to create a vacuum. You can put your hand on the throat or smallest diameter section and it will be cold, but put your hand near the discharge and it will be hot. All the energy is converted back into heat. There is no loss of heat in the whole system except for the little bit of radiation. Permanently cold air can be obtained only by making the expanding air do work and removing the work energy in the form of electricity or hoisted ore, as examples, from the mine.

M. Mosier.—It was an Ingersoll-Rand drill we were using. We noticed that the air current induced by differences in temperature had a maximum velocity of 60 ft. per min., when the drill was not running. When the drill was running, it did not increase the velocity of the air current. In fact, it decreased it. We assumed that the reason for this was a cooling effect which reduced the action of the normal temperature difference.

W. R. Chedsey,* State College, Pa.—This has a very close electrical analogy. I think most of us are sufficiently familiar with electrical calculations with symbols that we are used to so that I can contribute to the discussion in that way. A certain current flowing at a certain voltage through a resistance develops a heat or temperature effect that we are familiar with; I mean through a dead resistance. If the same current at the same voltage does work in a motor, very much less heat develops. The energy has occurred as mechanical work instead of as heat. Similarly with compressed air, except that instead of a heat condition there is a cooling condition, because at the start, before work was done with it, the air was at normal atmospheric temperature. If no work is done and the air is just allowed to expand there is practically no cooling effect. In other words, it has a tendency to cool, but to heat immediately exactly as the electrical resistance for heating would do; but when compressed air is made to do mechanical work—whether it is crushing rock in a rock drill or expanding in a turbine, or whatever it may be, it cools off and does not heat up.

T. T. Read,† New York, N. Y.—The difference between temperature and total heat of the air current must be kept in mind. The air current which is coming out of the mine may become hotter or colder, as the case may be, but it may not be removing any heat of the mine because there has been no change in the total heat it carries. The two different cases must be kept clearly in mind, because one involves a problem of heat transfer and the other is simply a change in the temperature of the air without involving heat transfer.

P. Bancel.—Just one more point that really should go in the record. When investigating the cooling of a mine with expansion of compressed air, doing work, do not overlook the water vapor in the air. The mathematics in this paper refer to dry air. As air containing moisture expands a point is finally reached where the water vapor starts to condense and does work like a steam in a turbine. These formulas for temperature no longer apply. You might calculate a 100° drop in temperature and get only 50° drop.

A book[1] by Ewing and Egan discusses conditioning with compressed air at three mines, one of them the Robinson Deep. Carrier cools on the surface and gets very

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† Vinton Professor of Mining Engineering, School of Mines, Columbia University.

[1] S. E. T. Ewing and A. L. Egan: Mine Cooling by Devaporized Compressed Air. Johannesburg, 1934. Radford, Adlington, Ltd.

dry air and pumps that down into the mine through one of the shafts. The York conditioning, as I remember, is installed at some low level, and the condensing water is pumped up. In the third mine, dry compressed air is put down and is made to do work in the mine.

T. T. Read.—What pressure does the third work on?

P. Bancel.—I do not remember offhand. Nothing unusual. I think it is the ordinary mine supply.

W. S. Weeks (written discussion).—In this paper I endeavored to work out analytically what would happen under various conditions. Perhaps it would be well to restate the facts as I see them.

1. If compressed air is allowed to expand through an orifice in an air line supplied by a running compressor, the air after it has come to rest will be the same temperature as when it started. While flowing through the orifice at high velocity it will be cold because at that point heat energy has been converted into kinetic energy. The work done by the compressor in pushing the air into the line (not in building up the pressure) is balanced by the work done by the air in pushing aside the atmosphere, so no heating or cooling results from these effects. The heat energy that has gone into velocity all goes back into heat when the air comes to rest.

2. If air from a line at constant pressure is allowed to flow into a closed receiver, the air after it comes to rest in the receiver will be warmer than when in the air line. This is because no external work is done in the receiver and we have the work done by the compressor in forcing the air into the line. This work must appear somewhere and it appears as heat in the receiver.

I first noticed this effect when studying the evacuation of tanks by vacuum pumps. When atmospheric air was allowed to flow into an evacuated tank a marked rise in temperature took place as shown by thermocouples.

3. If air is allowed to flow into a closed receiver and, before the temperature returns to the atmospheric temperature, the air is exhausted to atmosphere, the temperature of the air after coming to rest will be the same as it was in the air line.

If, however, the air after being admitted to the receiver is cooled to the atmospheric temperature and then is allowed to flow to atmosphere, its temperature after coming to rest will be lower than its temperature when in the receiver. This is because the air has to do work against the atmosphere, and since the receiver is cut off from the compressor this work must be done by the heat energy in the air.

4. If air is used at line pressure throughout the stroke of a direct-acting pump or rock drill, the entire work is done by the thrust of the compressor in forcing the compressed air into the line. No energy is removed from the air by the process. It was the work done by this thrust that caused the heating when the air flowed into the closed receiver. This work is now absorbed by the piston so the air fills the cylinder at the same temperature as in the air line. Conditions now are that of the receiver after it has been cooled to the atmospheric temperature. When the exhaust opens the air locked in the cylinder expands and heat energy is converted into kinetic energy. The air as it comes through the port is extremely cold, which accounts for freezing at that point. When it comes to rest, the kinetic energy goes back into heat but work has been done against the atmosphere and this work must come from the heat energy of the air so the air after coming to rest is colder than it was in the air line, but not as cold as when coming out of the exhaust.

This I think answers Mr. Mosier's question.

In regard to Professor Chedsey's contribution—we must distinguish between work done by the thrust of the compressor and work done by the adiabatic expansion of the air at the expense of heat. Work done in a rock drill operated without expansion

comes entirely from the thrust of the compressor. The medium from a theoretical standpoint might just as well be water.

No change of temperature takes place during the stroke. The absorption of this work by the piston merely keeps the temperature from rising.

He says, "If no work is done and the air is just allowed to expand there is practically no cooling effect."

If we expand from the air line we get no cooling, but if we expand from a closed receiver we get a large cooling effect.

In regard to the effect of moisture on temperatures as suggested by Mr. Bancel, it does not seem to me that the effect would be as great as he states. A cubic foot of saturated compressed air at 90 lb. per sq. in. gauge at sea level would weigh about ½ lb., if the temperature was 90° F. and the cubic foot would contain 0.002 lb. water vapor. If all of this condensed, liberating an average latent heat of say, 1070 B.t.u. per lb., it would heat the half-pound of air about 18°. If the air were saturated at 70° F. the condensation effect would not be over 10°.

May I express my gratitude to Mr. Bancel for presenting the paper and to the members for discussing it.

Control of Underground Mine Fires at Tintic Standard Mine

By Earl F. Hanson,* Member A.I.M.E.

(New York Meeting, February, 1937)

Fires in heavily timbered mines are disastrous, involving danger to both life and property. Some mines have been completely ruined or so heavily damaged that reopening them would not pay. Though few mines are entirely free from danger of fire, the risk is minimized by proper care and good design of plant.

The Tintic Standard mine is at Dividend, Utah, in the famous Tintic mining district, about 60 miles south of Salt Lake City. The development of the mine was begun in 1907 by the late Mr. E. J. Raddatz and his associates. From the beginning, the work was hampered by high rock temperatures and the presence of high concentrations of rock gases such as carbon dioxide and nitrogen. Development work was carried on with great perseverance for nine years, and resulted in the discovery of ore in 1916. The total production of the Tintic Standard mine to Jan. 1, 1936, amounted to 1,664,388 tons of ore, having a gross value of $61,996,719.

General Layout of the Mine

The No. 2 shaft, which is the main working shaft of the Tintic Standard, is within the main orebody and is connected with the five other shafts. Their respective positions with respect to the No. 2 shaft are as follows: North Lily, 2770 ft. northwest with drift connections on the 700 and 900-ft. levels; No. 3 shaft, 670 ft. east connected through the 700-ft. level; Eureka Standard, 4550 ft. south, connected on the 900-ft. level; No. 1 shaft, 1470 ft. southwest, with connecting drifts on the 900 and 1250-ft. levels; Eureka Lilly, 2000 ft. southwest, with connecting drift and raise through the 1250-ft. level.

The Tintic Standard orebody consists essentially of one continuous oreshoot extending from the 600-ft. level to the 1450-ft. level, with a cross-sectional area of approximately 70,000 sq. ft. at both top and bottom, reaching a maximum area of 215,000 sq. ft. between the 1100

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and 1250-ft. levels. See Fig. 2. Three small stopes, separated by pillars of ore composed of finely crystallized sulfides, extend upward from the 1450-ft. level to a little above the 1250 level, where they connect with a large stoped area surrounding the shaft.

Mining and Ventilation Methods

The hanging wall is composed of soft shale and altered limestone and the footwall is quartzite. Because of heavy ground, all of the stopes

Fig. 1.—Tintic Standard No. 2 shaft and surface buildings, with a part of the town of Dividend in the foreground.

have been timbered with square sets consisting of 10 by 10-in. posts and caps and 6 by 10-in. girts. The ore is mined progressively by taking out sections one or two sets wide, depending on the nature of the ground. Each section is lagged on the sides and filled with waste limestone or shale obtained from the development headings. Thus, the mined area consists of a mixture of timber and waste rock. It is estimated that there were approximately 30,000,000 board feet of timber in this mined area at the time of the recent fire. The timber throughout this area was connected.

The mine is in a high-temperature area. In virgin rock the average temperature is about 103° F. Ventilation is mechanical throughout. The average mine temperatures are 79° F. dry bulb, and 74° F. wet bulb, and there is a relative humidity of 79 per cent.

In the primary air circuit there are four connected shafts, one of which is the North Lily shaft. The circuit is controlled by four fans, handling about 76,000 cu. ft. of air per minute, and supplies air for the secondary circuits, consisting of 22 small fans, which pick up the air from the primary circuit and deliver it to the various working places.

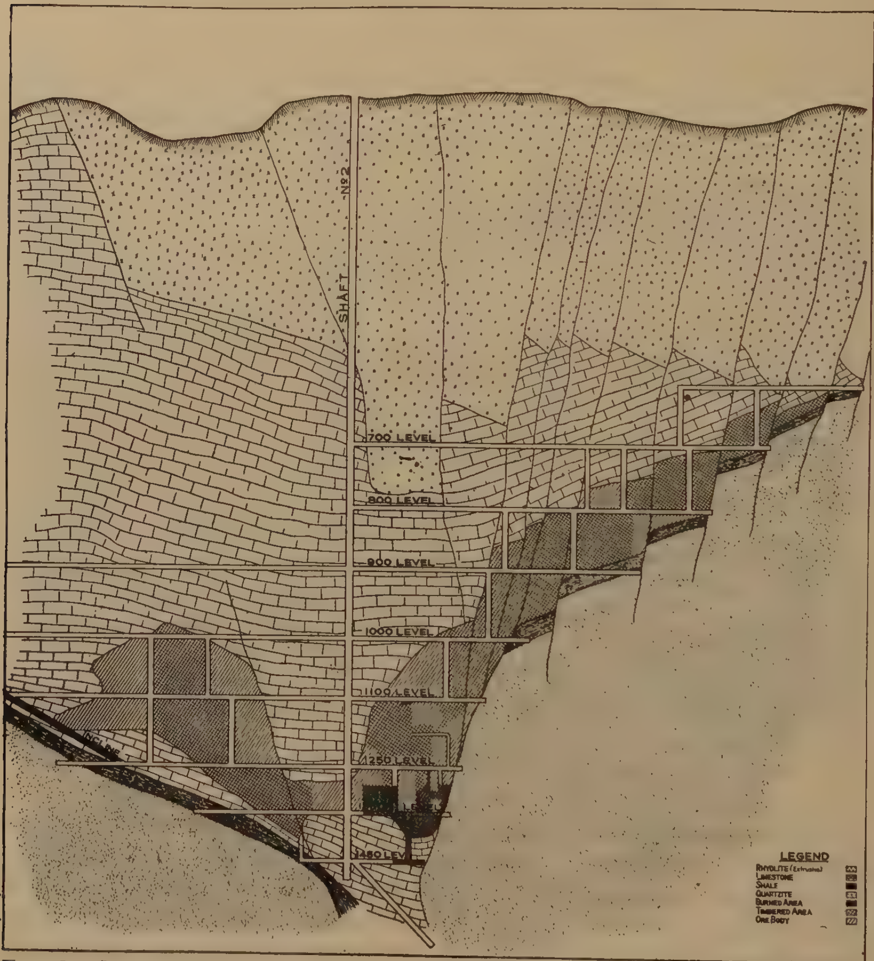

FIG. 2.—IDEAL SECTION ON LINE N. 45° 00′ E. THROUGH NO. 2 SHAFT, TINTIC STANDARD MINE.

The No. 2 shaft is downcast and is connected with all the main levels. It furnishes an ideal supply of air for the fans in the secondary system. Electrically operated doors are installed on the various levels to prevent recirculation. About 39,000 cu. ft. of air per minute are taken down the No. 2 shaft. Most of this air is used in ventilating the lower levels of the mine. It is then discharged to surface through the

No. 1 shaft by a 40-hp. double-inlet fan handling 31,500 cu. ft. per minute at the 1000-ft. level, and a 50-hp. single-inlet fan handling 23,000 cu ft. per minute on the 1300-ft. level

The North Lily shaft is also downcast. A 30-hp. single-inlet fan supplies 26,000 cu. ft per minute through the 900-ft. level connection to the Tintic Standard. The amount of air supplied from the North Lily through the 700-ft. level connection is variable, depending upon the amount of air needed to ventilate that section of the mine. The North Lily mine is in a relatively low-temperature area, so that the air, after passing through it, is still desirable both as to temperature and humidity. This air ventilates the northeast section of the Tintic Standard and leaves the mine by way of the No. 3 shaft, at the collar of which is located a 40-hp. single-inlet fan discharging 22,000 cu. ft. per minute.

System of Fire Protection

Extraordinary precautions are taken against fire, both on the surface and underground. Surface buildings are covered with corrugated iron sheeting and are situated as far from the shaft as is practical. Fire hydrants are placed near all the main buildings and near the shaft on each level of the mine. All hydrants are connected with fire hose wound on reels protected by steel covers. No smoking or open lights are allowed on the cages or near the shaft, either on the surface or underground. Electric safety lamps are used by employees working in the shaft or in closely timbered places where there is danger from an open-flame lamp. Within the next month, all underground employees will be equipped with these electric safety lamps.

The electric wiring which consists of armored cable in the shaft and heavily insulated cable on the levels, is under constant observation against damage from sagging timbers. Underground transformers and ventilation fans are some distance from the shaft and away from timbers.

Complete mine-rescue equipment and a crew of 85 employees trained in mine-rescue work are available for any emergency. General plans for handling of emergencies are also worked out in advance.

In a stope from the 1250-ft. level some years ago, when spontaneous combustion due to blasting during mining operations caused the ore to ignite and burn slowly, producing sulfur dioxide gas, the hot rock was soon cooled with water and removed from the mine by the prompt action of the mine-rescue crew, but the recent fire was in a rather inaccessible and practically abandoned area of the mine and presented a situation very different from any that had been anticipated. This fire had its origin near the bottom of the orebody between the 1350 and 1450-ft. levels, in what is known as the 1455 stope. This stope was mined and filled except for a manway and chute some 15 years ago. The stope is surrounded in part by finely divided low-grade primary sulfide ore, contain-

ing a total of approximately 15 per cent sulfur This sulfide ore is undergoing oxidization, resulting in local rock temperatures up to 143° F. Timber in contact with this rock becomes charred. In confined and unventilated areas, the heat resulting from the oxidization of the sulfides finally reduces the timber in the entire area to charcoal. There is always a strong sulfur dioxide odor present. By reducing the rock temperature with good ventilation the fire hazard is minimized.

Fire Control

On the morning of July 17, 1934, considerable sulfur dioxide gas was discovered coming from the top of 1455 stope on the 1350-ft. level. An investigation disclosed that a slab of ore had sloughed off the side of the manway, causing the air course to be practically cut off and resulting in spontaneous combustion, which rapidly spread through the timber in the filled stope. Our problem was to remove or extinguish the burning material. The mine-rescue team, equipped with oxygen breathing apparatus, immediately tried to reach the fire. As the only approach was the manway immediately over the fire, the heat was intense, and the attempt to fight the fire at close range had to be abandoned. The next method of attack was to confine the fire to as small a zone as possible by preventing the circulation of air through the burning stope. Brattices, made by guniting a steeltex-covered framework, were placed in all drifts as close to the fire as possible on the 1350 and 1450-ft. levels. This was partly effective. The gas then traveled up through the filled stope and along the hanging wall of the orebody and came out of the stopes on the 1250 and 1100-ft. levels. Additional brattices were constructed on those levels, in an effort to confine the rising fire gases, and a careful, though unsuccessful, study was made to determine where the air was being admitted to the fire. Because it would be a long and difficult process to confine the gases that were rising through the filled stopes, it was decided to try to prevent their circulation. To do this the ventilation system was altered in such a way as to bring air pressure against the rising gases, thus preventing any intake of air through leaks in the surrounding rock. This was accomplished by stopping the exhaust fan at No. 3 shaft and directing against the rising gases the pressure of air supplied by the 900-ft. level fan, which pulls air from the North Lily shaft. The pressure was equalized by allowing air to escape through by-pass doors on the 900, 1100 and 1250-ft. levels. Almost immediately the entire mine down to the 1250-ft. level was cleared of smoke. Adjustments were made to allow the confined gases to remain well above the position of the fire so that no air from above could possibly reach it. It was now apparent that the key to the control lay in developing sufficient pressure against the gases to confine them within the fire area, thus depleting the oxygen in that area and resulting in the accumulation of heavy carbon dioxide

gas. During this procedure the temperatures and gas tests taken through the brattices dropped from a maximum of 168° to 145° F. and the carbon monoxide content dropped from 1 per cent to zero, while the oxygen content decreased to just above 1 per cent. This was conclusive proof that the fire was out. No further trouble occurred during the following three months, except occasionally when the fan on the 900-ft. level, which furnished the pressure, stopped because of power interruptions, allowing the confined sulfur dioxide gas to rise into the stopes immediately above. Brattices were replaced by doors and all stopes were again producing ore except the one below the 1250-ft. level.

It was evident that something would have to be done in order to permanently eliminate combustion. On Oct. 18, an investigation was started, entrance being made through the brattice on the 1450-ft. level. It was found that the fire had burned out considerable timber and had followed down a raise along the air route, where it had been extinguished by the cutting off of the air supply. A small amount of air was allowed to circulate, with favorable result. After several days of testing, ventilation was extended into the burned area, so that repairs could be made and the open country filled. Unusual precautions were exercised to make sure that any remaining fire would be immediately discovered. First, two ventilation doors were built in front of the bulkhead to form an airlock, so that when the bulkhead was opened the fire area would remain sealed and yet provide easy access for men and materials. Temperatures and gas analyses were taken several times a day through all bulkheads so that the effect of the admitted air could be observed.

Three days after ventilation was established, there burst forth a volume of wood smoke on the 1100-ft. level. The entire area was promptly sealed, but it was apparent that the fire was now burning above the 1350-ft. level in an inaccessible filled stope. Temperatures reached a maximum of 178° F. with 5 per cent carbon monoxide present. Immediate precautions were taken to prevent contamination of the primary air circuit of the mine by installing fans and pipe lines to convey the escaping gas into the exhaust air stream.

From results, we were positive that our method of attack was the proper one. The problem was to make a perfect job of the plan already adopted. A study of the mine maps and knowledge of the inaccessible mined area indicated the probable course of the fire at this point. A mine-rescue crew equipped with oxygen breathing apparatus was put to work cleaning out and retimbering 100 ft. of old drift on the 1250-ft. level, which connected with a raise from the 1355 stope, where the fire was believed to be burning. By capping this raise, one of two outlets would be cut off. When the raise was reached a large volume of sulfur gas at a temperature of 162° F. made it almost impossible to work even for short periods. A plug made from a piece of brattice cloth filled with

sawdust was forced down into the raise, temporarily stopping the flow while the raise was capped with concrete. A decided improvement resulted. Work was then concentrated on reaching the No. 5 stope near by, known to be an outlet for the gas evident on the 1250-ft. level. When this objective was reached the same high temperature and large volume of gas was encountered, so that 6000 cu. ft. per minute of fresh air had to be supplied to this one working face in order that apparatus men could work even in 10-minute relays. Work here had to be abandoned when the fire increased because of air being admitted at this point. The entrance to No. 5 stope was resealed, bringing the fire at this point under control. It was found advisable to reinforce all brattices with 12-in. reinforced concrete walls, extending well into the ground on all sides in order to overcome the leaks known to exist in the loose ground adjacent to the brattices. An investigation was made of the fire zone, by going through the brattices and inspecting the immediate fire area. This investigation showed that the fire had reached the 1356 stope and was slowly burning just above the sill floor of the 1350-ft. level and only 80 ft. away from the main shaft. Should the fire reach the 1250-ft. level it would spread to the shaft and into every section of the mine, as it would be practically impossible to confine it in this broad stoped area. It was decided to make a horizontal cut on top of the 1356 filled stope and lay a concrete slab just below the 1250-ft. level where the two stopes joined. Work was started, but subsidence of the stope above caused this plan to be abandoned in favor of a vertical wall to be extended around the top of the 1356 stope on the 1250-ft. level against the solid ground on all sides except for a small horizontal cap where the two stopes overlapped, thus separating the timbering and completely confining the fire in the stope below.

Great difficulty was experienced in driving a drift along which the wall was to be built because of the large amount of matted timber in the fine filling material encountered. The exceptionally heavy ground made it almost impossible to keep this drift open. The work was done largely without apparatus by keeping a small fresh-air chamber at the face under sufficient pressure to hold back the surrounding fire gas.

Subsidence

Considerable subsidence was now taking place. The burned-out timbers caused the stope filling to consolidate, leaving a large inaccessible opening under the 1100-ft. level. The filling above the level began to run and immediately we began to pour waste rock into the hole, working as fast as possible to prevent caving. More than 2500 tons were dumped to fill this hole.

Apparently the filling made a seal on top of the fire, practically eliminating all the gas and materially reducing the temperature. Subse-

quent settling opened up a vent and this time the filled stope above the 1100-ft. level settled, forming a mat across a narrow neck just below this level. A great effort was made to make an opening through which waste

FIG. 3.—REINFORCED CONCRETE BULKHEAD ON THE 1350-FT. LEVEL SHOWING SLIME LINE AND TEST PIPE.

might be dumped, but it had to be abandoned because of heat and dangerous working conditions. However, an opening was made sufficiently large to admit slimes.

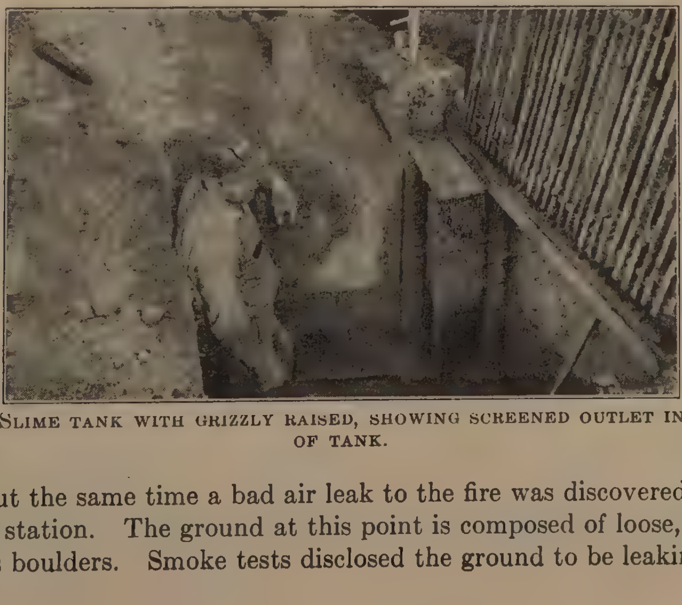

FIG. 4.—SLIME TANK WITH GRIZZLY RAISED, SHOWING SCREENED OUTLET IN CORNER OF TANK.

About the same time a bad air leak to the fire was discovered on the 1350-ft. station. The ground at this point is composed of loose, porous, siliceous boulders. Smoke tests disclosed the ground to be leaking for a

distance of 20 ft. on all sides of the main fire bulkhead. The porous ground was gunited but tests made through the bulkhead showed that oxygen was being admitted. On Jan. 19, 1935, a rubber tube inserted into this bulkhead through the test pipe burned off, indicating that the fire had followed up the admitted oxygen.

At this time the construction of the concrete wall on the 1250-ft. level was proceeding very slowly, because the ground was heavy and loose and because oxygen breathing apparatus was necessary during all the work. All brattices had been replaced by 12-in. reinforced concrete bulkheads (Fig. 3) but tests showed as high as 12 per cent oxygen behind

FIG. 5.—TUGGER HOIST AND SCRAPER AT WORK MOVING MATERIAL INTO SLIME TANK.

some bulkheads. It was obvious that a complete seal could not be made of the fire area and that something must be done to prevent further subsidence.

Slime

Sliming—that is, the filling of voids hydraulically—had been under consideration for some time, as it had been successfully applied in fighting fires at Butte. It had not been adopted previously in this fire because water introduced into the loosely consolidated limestones and shales of the Tintic Standard mine causes disintegration of the cementing material of the rocks, starts caving, and subsequently opens cracks for circulation of air.

Slime is a mixture of water and fine rock of a consistency that can be conveyed through pipe lines. About 40 per cent of water is necessary to keep the mixture running, and clay is the most satisfactory solid material.

A slime tank 5 by 5 ft. by 4 ft. deep, was constructed of 2-in. pine boards and set in the ground at the collar of the No. 2 shaft (Fig. 4). Agitation was accomplished by releasing compressed air through a manifold, connecting four rows of inch pipe having vents at 4-in. intervals,

set in the bottom of the tank. A double-drum electric tugger hoist pulling a scraper was set to move the material to the tank from the railroad track where it was dumped from cars and trucks (Fig. 5). A 2-in. pipe was installed in the shaft with valves and water connections for flushing on each level, whence laterals connected with the bulkheads.

Twenty-one bulkheads on the 1100, 1250, 1350 and 1450-ft. levels were strengthened to provide at least a 24-in. wall. Each bulkhead was fitted with a test pipe and a 2-in. slime pipe with a valve to control the discharge to the bulkhead. Another valve was placed in each lateral at its junction with the main line. Also, a valve was placed in the main line below each junction point. Thus all lines could be kept clean and allowed to remain empty when not in use.

A search was made for suitable material for sliming. First, fine sand was tried, shipped by rail from Jordan Narrows. The sand, however, quickly settled out, clogging the lines and building up close inside the bulkheads. This was overcome to some extent by adding 50 per cent clay soil. Finally, soft altered porphyry was used exclusively, with highly satisfactory results.

The country rock covering most of the surface in the vicinity of the Tintic Standard mine is rhyolite-porphyry of volcanic origin. This rock is locally decomposed by hydrothermal alteration and weathering, so that it has a fine soil-like texture for several feet in depth over the mineralized mine area.

This fine decomposed porphyry is easily mined. It is plowed with team and plow, then scraped by means of a double-drum air hoist pulling a scraper, which conveys it to a vibrating screen of ½-in. mesh. The reject, amounting to 10 per cent, is discarded and the screened material goes into a 5-ton truck, in which it is hauled about 1000 ft. to the stockpile at the collar of the shaft. This fine decomposed rhyolite-porphyry forms an impure concrete of considerable strength when allowed to set after being deposited as slime, so that loose or broken country rock adjacent to the stopes, as well as the stope filling, becomes strong and solid after sliming.

Sliming was started on March 30, 1935, by running one hour at a time through the various walls on the 1100, 1200, 1250 and 1350-ft. levels. In a short time the air leaks were stopped by the slime, the temperatures dropped, the sulfur dioxide gas subsided and no further trouble resulted. See Table 1. Sliming was continued, however, in order to completely fill the burned area and prevent any further subsidence; 10,000 tons of material were introduced at a maximum rate of 20 tons per hour through the 2-in. pipe, before it was worn enough to require replacement.

To date, more than 11,500 tons of porphyry, conveyed as slime, has been poured into the mine below the 1100-ft. level, with the result that

Table 1.—*Change in Temperature and Composition of Fire Gases during Fire Control*

Analysis at 1355A Bulkhead

Date 1935	CO, Per Cent	CO_2, Per Cent	O, Per Cent	SO_2, Per Cent	Temp rature, deg. F.	Pressure
Nov. 2	3.0	2.4	16.3		165	Positive
Nov. 4	1.	3.2	14.5		152	Positive
Nov. 6	0.5	4.1	12.6	1.36	146	Positive
Nov. 8	0.3	6.4	10.3		138	Positive
Nov. 10	0.2	7.0	8.0		134	Positive
Nov. 15	0.2	10.5	3.0		129	Positive
Nov. 30	Trace	11.8	1.7		124	Positive
Dec. 14	Trace	13.0	0.8		119	Positive

even the ordinary subsidence has been almost completely overcome, effecting a considerable saving in the usual maintenance cost of the mine.

Some apprehension was felt concerning the effect of introducing such a large volume of water into the fire area. So far, no damage has resulted. Tests of the humidity show that about 20 tons of additional water per day is leaving the mine in the air stream. The remaining water is apparently draining to the water level.

Instruments and Equipment

Instruments and equipment played an important part in the fire work, especially the carbon monoxide detector, the smoke tube and the all-service gas mask manufactured by the Mine Safety Appliances Co., as well as the portable Orsat gas-testing apparatus manufactured by the Denver Fire Clay Co. Three types of oxygen breathing apparatus were used, Gibbs, McCaa and Paul, and all gave satisfaction. In all, 1083 man-shifts of work were done with apparatus.

Accident Record

During the course of fire-control work, extreme precautions were taken to prevent illness and accidents. Fire work did not materially affect production except during the month of December, 1934, when about 40 per cent of the underground men were employed in building reinforced concrete bulkheads. Only two man-shifts were lost owing to accidents resulting from fire work, against a total of 4609 man-shifts charged to the fire work. Comparatively few cases of illness were reported.

Conclusions

1. It was proved that pressure control that confines the products of combustion within the fire area, depleting the oxygen supply, will stop the progress of a fire.

2. Bulkheads should be as close to the fire as possible and should be well bonded to the ground on all sides to prevent leaks. If the sealing is perfect, combustion is soon extinguished in the enclosed area by depletion of oxygen. Perfect sealing is impossible. Though the bulkheads may be airtight, air percolates through the fractured and porous strata. Bulkheads used in connection with sliming must be strong enough to withstand the hydrostatic pressure developed.

3. When combustion stops, the heat of a large fire surrounded by its ashes dissipates very slowly, thus making the opening of a fire area extremely hazardous even several months after the fire has been brought under control.

4. Water, when used, caused rapid disintegration of hanging wall and country rock, causing caves to develop. The slime mixture does not do this.

5. The cost of doing work with helmet crews is approximately three times that done by men working in fresh air. Therefore, considerable ventilation equipment and work is justified to eliminate the use of oxygen breathing apparatus.

6. A phenomenon observed was that when an area near a given bulkhead has apparently become filled or blocked with slimes, it may yet admit slimes from another area through a different bulkhead.

7. Only comparatively few men make efficient underground fire fighters. Just because a man has had mine safety training, it does not necessarily mean that he will make a good fire fighter.

8. The employing of experienced men, to check on all operations and to put into operation all that has been learned from other mine fires, is endorsed.

9. Fire fighting is a strenuous, hazardous undertaking and a job on which consistent, steady progress must be made in order to protect life and property. Mine fires do occur even where unusual fire-prevention measures are used, and for that reason every mine in which a number of men are employed should have fire-fighting equipment and fire-control plans ready for any emergency.

Acknowledgments

The Tintic Standard is indebted to the Bureau of Mines through Mr. D. J. Parker, Health and Safety Station, Salt Lake City, for rendering all service possible. Mr. Walter W. Kessler, of the Bureau of Mines, rendered excellent service in helping to care for the mine-rescue equipment that was so generously furnished by the Bureau. We appreciate the cooperation of Mr. E. A. Hodges, State Mine Inspector. We recognize the good work of Mr. John Hodge, William Trudeau and Fred Meire, supplied by the Anaconda Copper Co. of Butte, Mont. Mr. Hodge had

charge of the underground work and a great deal of credit is due him for his efforts and practical suggestions.

The writer wishes to acknowledge the fine cooperation given him in the fire-control work and in the preparation of this paper by Mr. James W. Wade, General Manager and Vice President, Mr. L. R. Dobbs, General Superintendent, and Mr. F. W. Hanson, Assistant Superintendent, of the Tintic Standard Mining Co.; also the many helpful suggestions received from Prof. R. S. Lewis, of the Mining Department at the University of Utah, during the preparation of this paper.

DISCUSSION

(*Daniel Harrington presiding*)

D. Harrington,* Washington, D. C.—The use of slimes for the handling of a mine fire is of particular benefit to metal mines because suitable conditions are far more likely to occur in metal mines than in coal mines. Metal-mine fires are more likely to be difficult to handle than coal-mine fires.

The slimes are the last resort, but where used intelligently, they will do the "job." Slimes have been used in a number of metal-mine fires in the United States, probably the outstanding case being that of the Anaconda Copper Mining Co. in Butte. The fire had extended down around the 2000-ft. level and up close to the surface, around the 300, 400 or 500-ft. level, extended over a considerable area, and not only tied up a lot of ore immediately, but also had possibilities for extending, and was extending very rapidly and rather alarmingly, and it might have practically ended the life of much of the valuable ores of the Butte district unless something of a radical nature was done. I had the very valuable experience of being there while they were fighting the fire with numerous methods, and at the time when they finally decided on the use of slimes.

They were confronted with problems somewhat similar to those at the Tintic. They had high rock temperatures; and fires in back-filled timbered regions. It is a fallacy to think that it is impossible for fire to get into timbered regions which are back-filled with small material. A fire can get in there in many ways and when it does, it is about the toughest fire on earth to handle, because no matter how tight the back-filling, there are likely to be places where the air can get through, and if the air can get through, a fire will extend itself. That is what happened in Butte and that is what happened here. They had broken territory in Butte as they apparently had here. They tried water, they tried sealing; in Butte hundreds of thousands of dollars were spent trying to seal with bulkheads to restrict the air and it could not be done. They drilled hundreds of thousands of linear feet of holes with diamond drills and forced water in every conceivable manner from the surface, from various regions underground, at various angles, horizontal, even up as well as down. The water would find a course, and later on they found places where it had been running within a foot or so of burning timber and did practically no good. They finally resorted to slimes and the Anaconda Copper Mining Co. put into this one fire more than 1,000,000 tons of solid material in the form of slimes, or about 5,000,000 tons of water and solids combined. As at Tintic, they had to get the right type of material, the right size, and after much experimental work, they used the slimes from the Butte and Superior mill. The mixture was about 3 or 4 parts water to 1 part slime or rock.

They have at least controlled the fire. It is pretty well extinguished in many ways, but they still have very high temperatures, and I understand (I have not been

* Chief Engineer, Safety Division, U. S. Bureau of Mines.

there for 10 years) that when they do go into this territory they have to "watch their step" for fear of re-ignition from residual heat.

There are several papers on the use of slimes in mine fires. One of the best of them is by H. J. Rahilly[1] who is now the assistant superintendent of the Anaconda Copper Mining Co. He had general charge of fire fighting with slime filling in Butte for many years. Another is by Oscar A. Glaeser,[2] in connection with the United Verde Mine in Arizona, where they had an experience somewhat similar to that at Butte.

S. Veazey,* Butte, Mont.—Much of that same section is being mined again, but so far as I know, is not giving trouble. Present practice is to bulkhead danger areas before trouble develops, placing bulkheads in the crosscuts giving access to the orebody. Two light concrete walls are erected about 10 ft. apart in a crosscut, and the space between filled solid with slimes. Then if the ground settles and the concrete walls do not hold, the weight falls upon the slime filling and the seal is not destroyed. Old workings down to present operating levels are being sealed off in this manner, preparing them for filling at any time fire should break out.

D. Harrington.—Has much of the fire territory been worked?

S. Veazey.—Yes, it has; they are back into the worst of it. The fire is largely out.

D. Harrington.—Have you worked the fire territory down as far as the 2000-ft. level?

S. Veazey.—Yes, they had to fill clear down to 26 and 28. They are back into that now.

B. F. Tillson,† Upper Montclair, N. J.—I would like to have someone outline the specific function of the slime as compared to water or loam or something else, as a sealing agent, the factor it scientifically plays in comparison with the other.

D. Harrington.—All I can tell you is this, Mr. Tillson: Water was used to the *n*th degree in Butte, and the difficulty was that the fire territory was very badly broken and the water filtered or seeped away before it had any effect on the fire.

With the slimes results differed as to place. The procedure was to diamond-drill into the fire territory—partly by guess, partly locating the drill holes by surveys. When the drill holes were in open territory, slimes were introduced under pressure. The pressures were fairly high, sometimes around 200 lb. The slimes would run for varying periods into this territory; sometimes for a day or two or one, two or three weeks, but ultimately the end of the hole into the fire territory would choke. I suppose that the combination of water and mineral material came into more or less intimate contact with the fire. The water would evaporate and the solid material would build up. In some places there was difficulty in effecting this, the actual filtering out of the solid material probably taking place only when the water was stopped and allowed to back up and gradually filter or settle out mechanically. At any rate, they did fill the fire territory and five, six or more years later they found the slime material so compact that in some places it was necessary to drill in order to get to the ore pillars involved. And by the way, the estimated value of the ore involved in these pillars ran as high as $12,000,000.

G. Sherman,† New York, N. Y.— Some years ago I saw some of the Butte fire country and I have a theory which I would like to contribute. When water is poured

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[1] Mine Fires and Hydraulic Filling. *Trans.* A.I.M.E. (1923) **68,** 61–76.

[2] Ventilation at United Verde Mine. *Trans.* A.I.M.E. (1929) **85,** 114–143.

* Research Engineer, Anaconda Copper Mining Co.

† Consulting Engineer.

into a vein with a pitch, it flows down, as nearly vertically as possible, to the footwall and then follows on the footwall. The vein fill above, broken waste and timbers, gets little or none of it. Dry fill would slow down the fire to a smouldering mass that may hold it dormant for a long time, but it is water that puts it out. A stream of slime or fine sands builds up until it blocks itself and then breaks out in another direction until every open space is filled with a wet quaking, practically solid mass, and every timber and rock is surrounded. The slime is thus a means of distributing the water throughout the mass. It was noted in a reopened fire section in Butte that the slimes had packed themselves as solidly under the charred timbers as above. The consistency of the slime is the controlling factor in its successful use.

Determination of Effectiveness of Dust-control Measures Used in Mines

By J. William Fehnel*

(New York Meeting, February, 1935)

The health hazard due to the inhalation of certain dusts in the mining industry has been recognized, but was not subject to state or federal inquiry in the United States until about 1916. Reports on miners' phthisis and associated effect of dust inhalation were published as early as 1903 in South Africa and in 1906 in Australia. A report was made by Lanza[1] in January 1917 of a study of 433 cases of miners' consumption among zinc miners in southwestern Missouri. In a preliminary report issued in 1915 Lanza and Higgins[2] concluded that, while there were general causes, such as housing and working conditions, that tended to produce a high incidence of tuberculosis, the flint dust constituted an unusual element of danger, to which it was reasonable to attribute the unusual prevalence of tuberculosis. Later a clinic established under the joint sponsorship of the United States Bureau of Mines, the mine owners, and the Metropolitan Life Insurance Co. carried on an extensive study covering a period of five years. The results of this study have borne out the conclusions reached in the earlier surveys.

The early surveys in this country indicated the necessity of determining the extent of the dust pollution of air associated with the various mining operations. The first determinations were reported on the weight basis, undoubtedly to make comparisons with work previously done in Australia. Later studies of dust in industry led to the development of more refined methods of sampling and enumeration on the basis of count in place of weight, so that by 1919 Knowles[3] reported 53 methods.

Manuscript received at the office of the Institute Feb. 4, 1935.

* Chemist, Industrial Hygiene Laboratory, Metropolitan Life Insurance Co., New York, N. Y.

[1] A. J. Lanza and S. B. Childs: Miners' Consumption, a Study of 433 Cases of the Disease Among Zinc Miners in Southwestern Missouri. U.S. Public Health *Bull.* 85 (1917) 32.

[2] A. J. Lanza and E. Higgins: Pulmonary Disease Among Miners in the Joplin District, Mo. and Its Relation to Rock Dusts in the Mines. U.S. Bur. Mines *Tech. Paper* 105 (1915).

[3] E. R. Knowles: Dust Determinations in Air and Gases. *Trans.* Amer. Soc. Heat. and Vent. Engrs. (1919) **25**, 101–132.

In 1925 the United States Public Health Service[4] issued a report on comparative tests of instruments for determining atmospheric dusts, the results of a study started in 1922. The United States Public Health Service and the United States Bureau of Mines finally standardized upon the impinger method, which they have been using and describing in their various published reports of dust surveys for the past 10 years. The impinger method is described in detail in United States Public Health Reports (1932) **47,** No. 12.

Health Hazard and Its Prevention

The extent of a dust health hazard is dependent upon the following factors: (1) concentration of dust; (2) dust-particle size; (3) chemical composition of the dust; (4) length of time of exposure of the workmen; (5) possibly, individual susceptibility.

After it has been established that a dust health hazard exists, the next step is to correct the conditions so as to eliminate or control the dust condition. This may be accomplished as follows: (1) changing processes; (2) using other raw materials; (3) providing dust-allaying media; (4) entrapping and removing dust at the source of generation; (5) providing each workman exposed with individual protection. The first two considerations are not applicable to mining operations. Respirators are not only unsatisfactory but have a tendency to cut down the workman's efficiency, and it is almost impossible to get men to wear them constantly. Respirators have their place but only as a last resort.

Allaying dust by means of water was suggested by the early investigators as the solution of the dust problem. While wet drilling does materially allay the dust, still in dusts of very high silica content the concentration is often not lowered sufficiently for safe breathing. The use of water in drilling not only means that the drillers work in a damp atmosphere but the water is discharged, wetting their clothing and often covering them with sludge.

Various oils and foams have been tried at times but no successful tests have been recorded.

The entrapping and removing of the dust at its source of generation is the most desirable solution. A system to accomplish this purpose for mine-drilling operations must be flexible so that the apparatus is adaptable to all angles of position of operation. A sufficient air flow is necessary to pick up and carry the dust away from the source, and the conductors in turn must not only be flexible but must not be cumbersome, so that they can readily be handled and installed. The distance the dust is conveyed before it is filtered out of the air stream is also limited, owing principally to function losses in the conductor.

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[4] Comparative Tests of Instruments for Determining Atmospheric Dusts. U.S. Public Health *Bull.* 144 (1925) 41.

The mechanism for catching and removing the dust from the air stream must be capable of handling a heavy concentration and of storing up a large quantity of entrapped dust, and it must have a high filtering efficiency on the smaller sizes; i.e., those less than 10 microns. The impinger dust-sampling method is valuable in determining this efficiency.

Dust Counts

A series of tests was made to determine the extent of dustiness under different mining conditions in the Balmat mine of the St. Joseph Lead Co. near Gouverneur, N.Y. Atmospheric dust samples were taken at the breathing levels of the workmen operating the drills. A sample was also taken to determine the normal dust content of the air entering the mine location being worked. Table 1 shows the dust counts in millions of particles less than 10 microns in longest diameters. Attention is called to the severe conditions under which these tests were conducted.

Table 1.—*Dust Counts in Drifting on 700-foot Level*

Sample No.	Operation	Dust Counts per Cu. Ft. Air, Millions of Particles Less than 10 Microns	Remarks
15	Dry drilling with two drifters	266.7	Sampling started 5 minutes after drills were started.
1*a*	25 ft. away from 700-D2 face; wet drifting, two drills operating	6.4	General air sample, while taking samples 2*a* and 4*a*.
2*a*	At 700-D2 face; wet drifting; two drills operating	14.0	
4*a*	At 700-D2 face; wet drifting; two drills operating	11.9	Check on sample 2*a*.
3*a*	230 ft. from 700-D2 face, during wet drifting; two drills operating	5.4	General air while taking samples 2*a* and 4*a*.
5*a*	Kadco drilling at face	3.4	
8*a*	Kadco drilling at face	3.2	Check on sample 5*a*.
6*a*	50 ft. away from face	4.6	General air sample while taking samples 5*a* and 8*a*.
7*a*	230 ft. from face	2.9	General air sample while taking samples 5*a* and 8*a*.
9*a*	New type of hood, 230 ft. from face	2.9	General air sample while taking samples 11*a* and 12*a*.
10*a*	At Kadco exhaust	2.0	Sampled during total period. of Kadco drilling.
11*a*	Kadco drilling at face, new type of hood	2.6	
12*a*	Kadco drilling at face, new type of hood	2.4	Check on sample 11*a*.

The mine is very dry, not more than 50 gal. of water per minute is handled. At both locations drilling was done in dry rock.

The test on Kadco controlled drilling was a regular run for blasting. The two drills had been in operation for 1½ hr. before the test was made. Blasts were set off in other drifts on this level during sampling. It

TABLE 2.—*Dust Counts while Stoping in Raise on 500-foot Level*

Sample No.	Operation	Dust Counts per Cu. Ft. Air, Millions of Particles Less than 10 Microns	General Air-dust Count	Remarks
11	Dry stoping	208.0	9.1	
9	Wet stoping	9.9	7.1	Sample taken after 15 minutes of drilling.
8	Dry stoping with Kadco dust control	1.7	24.0	Sample taken after 15 minutes of drilling; blasting done on 700-ft. level during test.

was noted that a cloud of smoke passed up to the face of the drift where the sample was being taken. No doubt this air flow to the face was due to the effect of the Kadco exhaust. All these tests were made in D2 drift on the 700-ft. level. Table 2 shows the counts on the 500-ft. level while stoping on D2 raise.

TABLE 3.—*Dust Count while Mucking on 700-foot Level, D2 Drift*

Sample No.	Operation	Dust Counts per Cu. Ft. Air, Millions of Particles Less than 10 Microns		Remarks
1	General air D2 drift 700-ft. level	2.0		Sampled before operations began and before taking of samples 3 and 4.
2	General air at 700-ft. level station	1.6		Sampled before operations began and before taking of samples 3 and 4.
4	Dry mucking—sampled at left side of car	76.3	Average 82.8	Two men shoveling; sample taken 30 minutes after mucking started.
3	Dry mucking—sampled at right side of car	89.3		Two men shoveling; sample taken 30 minutes after mucking started.
5	General air—700-ft. level station	1.5		Sample taken 30 minutes after starting to muck and taking of samples 3 and 4.
13	Mucking with exhaust on car	8.0		Two men shoveling.
14	General air	3.9		Sampled during mucking and while taking sample 13.

There was a noticeable air current through the drift. Blasting was done on the 700-ft. level, while these tests were made, and undoubtedly

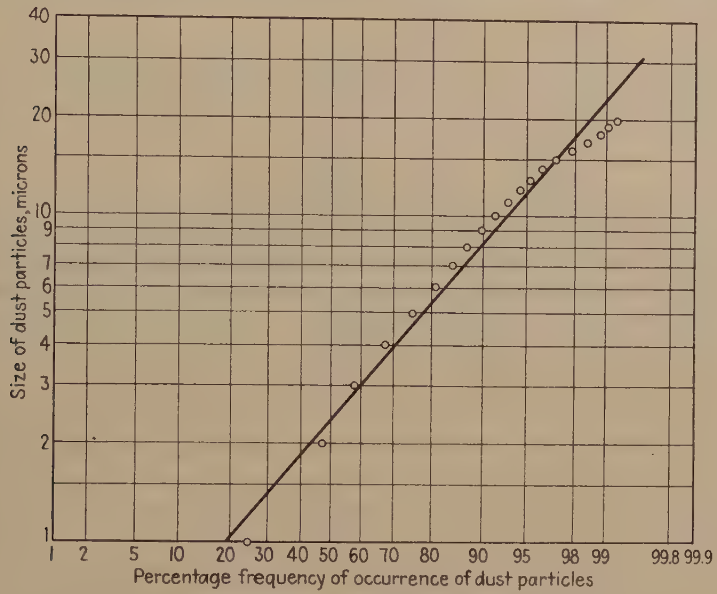

FIG. 1.—AIR SAMPLE TAKEN DURING WET DRILLING.

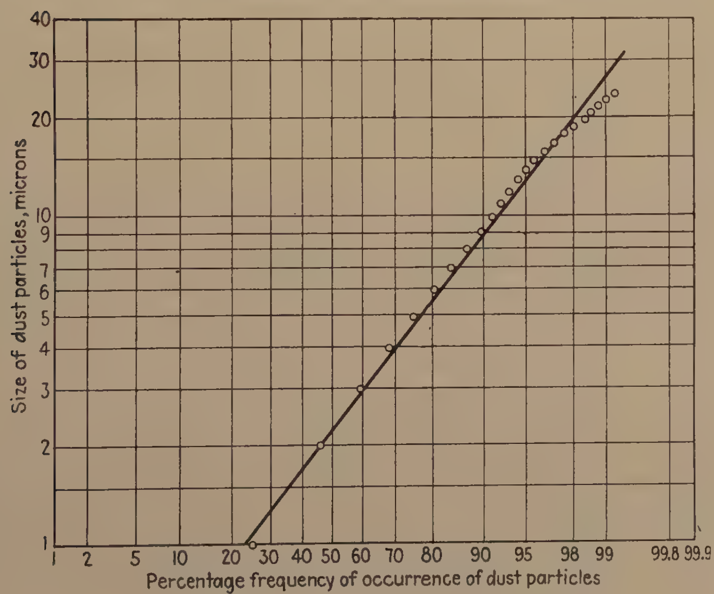

FIG. 2.—SAMPLE TAKEN DURING DRY DRIFTING.

accounts for the high count obtained for the general air taken at the same time as the test with Kadco controlled dry drilling.

A test was made of dustiness during mucking. The blasted material is not wet but mucked dry. The heaviest dust clouds were set up when the material was dropped into the mine car.

A sheet-iron hood was placed at one end and on the top of the car and connected with galvanized leader pipe to the Kadco machine, placed about 200 ft. back of the mucking operation. While there was considerable leakage in the conductor pipe and the machine had a capacity of only 300 cu. ft. of air per minute, a marked improvement over dry mucking was noted. Table 3 shows the dust counts for these operations.

Herewith are two graphs (Figs. 1 and 2) showing particle sizes of air-borne dust collected at the time of dry drilling and during wet drilling. It is interesting that the range of particle sizes is the same; i.e., wet drilling shows a median of 2.4 microns and 93 per cent less than 10 microns, while dry drilling shows a median of 2.2 microns and 92 per cent less than 10 microns.

Some Fundamental Data on Mechanical Dust Traps

By Theodore Hatch*

(New York Meeting, February, 1935)

The pneumatic rock drill operates essentially as a crusher. The rock is shattered into a powder that is removed from the drill hole by the air stream introduced through the hollow drill steel. Particles thus produced vary in size from ¼ in. or larger down to diameters too small to be revealed by the ordinary microscope. The power consumed by the drill is proportional to the new surface area of the powdered material, therefore the greater the proportion of "dust" produced, the higher the power consumption and the lower the drilling speed. A considerable amount of material ejected from the hole settles on the rock and much of this falls back as the mound builds up to be crushed further before it is removed again. The cushion of dust at the bottom of the hole reduces the effectiveness of the blow of the bit and thus lowers the rate of drilling. The unfavorable working conditions created by the dust also reduce the drilling speed by lowering the efficiency of the operator.

Thus, the dust produced by rock drills constitutes a serious drawback to economical performance on all drilling operations. To the economic objections, one must add the more important objection to the health hazard created when the dust contains free silica (quartz). The silicosis problem is a familiar one in many mines, tunnels, and other underground operations as well as in quarries and other types of open rock excavation.

Dust is a necessary end product of all pneumatic drilling operations. Its production cannot be eliminated although the proportion of microscopic particles can be reduced by using improved drill steel, proper methods of sharpening, more efficient drills, etc.—improvements that increase the drilling speed largely by increasing the size of the cuttings. The absolute improvement secured in this way is not great, however, and the pneumatic drill must always remain a dust-producing machine. Definite benefits do result, on the other hand, from the use of dust-control equipment that captures the dust as it emerges from the hole and conveys it to a remote point of discharge. Dust *production* continues, but dust *dispersion* into the atmosphere is prevented. The dust is removed as rapidly as it is formed, with the following results: (1) The silicosis

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hazard is reduced; (2) general working conditions are improved; (3) dust does not fall back into the hole to jam the drill steel and reduce drilling speed.

Requirements in Design of Equipment

The required apparatus consists of four major parts: (1) exhaust hoods, (2) suction piping, (3) air-cleaning plant and (4) source of suction. These must be designed to perform their individual tasks but at the same time they must fit together to provide a well balanced system of control. The plant may be of the unit type, i.e., self-contained apparatus for each drill, or in the form of a central installation with branches to the several drills, depending upon whether the drilling operations are scattered or concentrated. It may also be designed as portable, semi-portable, or stationary equipment.

The design is subject to certain practical limitations as well as minimum performance requirements with respect to dust control. Practical limitations are: (1) It must not interfere with the operation of the drill; (2) equipment must be flexible and capable of easy adjustment and change as the work progresses; (3) minimum attention should be required to maintain and operate the system.

Hygienic Requirements

Silicosis and other disabling pneumoconioses do not develop when the dust concentration in the air breathed by those exposed is maintained continuously below a certain permissible level—the so-called threshold dose. It is the function of the dust-control equipment to secure this result.

The safe concentration varies with the kind of dust, its physical and chemical nature, and particularly the content of free silica or quartz. Permissible amounts are not known for all materials and the standard to be employed in a particular case must be regarded only as an estimate which carries no guarantee of full protection for *all* the workers exposed. Threshold concentrations have been established in five notable industries: the South African gold mines, sandstone quarrying, tunneling and dressing, granite cutting and coal mining. These values are not directly comparable since they were not arrived at in the same manner, but a consideration of these and other data indicates that Table 1 may be accepted as a reasonable tabulation of minimum requirements for dust control. These figures must not be accepted as final. The safest practice is to consider all dusts guilty until proved innocent and, in the absence of definite data, to adopt as an arbitrary standard a figure always below the minimum dust concentration existing in the best factory or other unit of the industry. In this way, the industry as a whole is simply

asked to accomplish what has already been done in one or more units of that industry.

TABLE 1.—*Safe Dust Concentrations. Impinger Samples; Light Field, Low Power Counts*

Kind of Dust	Maximum Permissible Concentration, Millions per Cubic Foot
Containing no silica	50
Containing any small amount of silica, free or combined	30
Containing 20 to 40 per cent free silica	10
Greater than 40 per cent free silica	5 and preferably as much less as possible

Hood Design

Dust is produced by the rapid blows of the bit at the bottom of the hole, from which it is removed continuously by the stream of air discharged through the hollow drill steel. The rate of airflow varies with the air pressure, diameter of the bore in the steel and the depth of the hole; 30 cu. ft. per minute may be taken as a high figure. Assuming a hole diameter of 2 in. and 1-in. round steel, the air velocity out of the drill hole is 1800 ft. per min. Expansion occurs when the dust-laden air emerges to the free atmosphere and the dust is dispersed rapidly in all directions. As the velocity drops much of the dust settles out—some at the drill hole into which it falls to be reground, but the fine and dangerous material remains in suspension and is widely disseminated. The function of the exhaust hood is twofold: first, it prevents the expansion and escape of the dust-laden air; second, the dust is guided into the exhaust system, through which it is removed continuously without clogging the hood or suction pipe.

The drilling operation permits the use of a surrounding hood at the point of entrance of the drill steel into the rock. The Kelley trap is of this type although a mechanical seal is not maintained either at the rock surface or around the drill steel. Air in considerable excess of the amount escaping through the drill steel is drawn into the hood, thus insuring an inward velocity at all openings great enough to prevent the escape of dust. Within the hood a dynamic wall of live air surrounds the volume of dust-laden air as it emerges. Three things are accomplished:

1. Continuous inward flow of clean air constricts the dust-laden air and causes it to flow toward the hood outlet.

2. Inward air currents block the escape of particles, which tend by virtue of their own kinetic energy to be projected out into the general atmosphere.

3. Dust-laden air is diluted with clean air and sufficient velocities are maintained at all points to prevent settlement and clogging.

The required rate of air flow through the Kelley trap depends upon its design as well as upon the type and size of the drill. In order to maintain the dustiness below the standard of safety for silica-bearing dust, the rate varies from 60 to 100 cu. ft. per minute for the Jackhamer, up to 200 cu. ft. per minute or more for larger drifters.

Exhaust Piping

The purpose of the suction piping is threefold: (1) to connect the various dust traps to a common source of suction, (2) to maintain the proper flow from each hood, and (3) to serve for the pneumatic transportation of the collected dust to the central air-cleaning plant. Connections to the hoods must be flexible and reasonably portable and for this purpose flexible hose is usually employed; long lines on semipermanent installations should be of metal and securely fixed in place. Minimum velocities for the transportation of dust vary with the size and density of the particles to be conveyed, as Dallavalle's equations show:

$$V = 13{,}300\frac{S}{S+1}d^{0.57} \qquad \text{(vertical lift)}$$

$$V = 6{,}000\frac{S}{S+1}d^{0.4} \qquad \text{(horizontal transportation)}$$

In general a velocity greater than 3000 ft. per minute is required for conveying dust from rock drills. A $2\frac{1}{8}$-in. hose has an area of 0.0245 sq. ft. and with a total flow of 80 cu. ft. per minute through a Jackhamer hood, the velocity is 3200 ft. per minute. It is desirable from the standpoint of power consumption to use larger hose for greater rates of flow, but the portability of the small hose may overbalance the added power cost.

Air-cleaning Plant

The design of equipment for removing dust from the air is determined in the main by three factors: (1) amount of dust to be collected and the loading per unit volume of air; (2) nature of the material to be handled; (3) efficiency of collection required.

Amount of Dust Collected.—The rate of dust production varies with the speed of drilling and the size of the hole being drilled and is quickly calculated from these two values. The Jackhamer, drilling 12-ft. holes, 2 to $1\frac{3}{8}$-in. in diameter, at the rate of 100 ft. a day, produces dust at the rate of 0.7 lb. per minute (assuming a specific gravity of 2.5). The dust loading in the air brought to the collector, assuming 80 cu. ft. per minute through each trap, is therefore 58 grains per cubic foot. The total amount of dust collected per drill-day is 240 lb., which occupies a volume of approximately 3 cubic feet.

Size of Dust Particles.—The dust particles vary in size from ¼ in. or more down to submicroscopic material; the actual distribution depends upon the nature of the rock and the speed of drilling. A typical screen analysis of disintegrated granite is shown in Table 2.

TABLE 2.—*Approximate Screen Analysis of Rock Dust Collected from Pneumatic Rock Drills*

Mesh Size	Percentage by Weight Passing	Mesh Size	Percentage by Weight Passing
20	97.3	120	62.5
40	91.3	150	57.8
60	77.6	200	47.3
80	74.5	325	36.2
100	63.8		

More than 35 per cent is less than 40μ in size (325 mesh) and of this at least 5 per cent is less than 10μ. In the latter case, the *number* of particles is enormous even though the percentage by weight is low. In one test it was found that over one million million particles (10^{12}) smaller than 1μ are produced during one minute of Jackhamer operation.

Efficiency of Collection.—The dustiness at the point of discharge from the collector must not be great enough to create a health hazard. Above ground the permissible concentration at this point is higher than in mines, since there is immediate dilution and dispersion, whereas underground the air is recirculated and must never contain more than the permissible amount. For rock high in silica this may be set at 5 million particles per cubic foot of air, which is equivalent approximately to 0.5 mg. per cubic meter for particles having an average size of 0.8μ. Hence, with an initial loading of 130,000 mg. per cubic meter (58 grains per cu. ft.), the over-all efficiency must be 99.999+ per cent—a figure that has no meaning. *Operating requirements must be based upon the amount of dust escaping, not upon the amount retained.*

Plant Design.—From the practical viewpoint, the air-cleaning equipment must be compact and as light as possible. It should have a dust-storage capacity for at least 4 hr. of drilling and should not require cleaning more often.

The heavy loading of highly nonuniform dust requires the removal of the bulk of the material (90 per cent and preferably 95 per cent) before filtration, in order to maintain the filter load at a level that does not produce excessive filter resistance. A special primary separator is required therefore, since a cyclone type of collector cannot be employed to remove particles as small as 5 to 10μ.

For best results, the velocity of filtration should not exceed 2 to 3 ft. per minute. Higher rates increase the resistance and cause the filter to

"break"; i.e., dust escapes through the cloth pores. The frequency of cleaning is also increased. A filter area of at least 30 sq. ft. per drill of the Jackhamer type is required and a simple and positive cleaning mechanism must be provided.

The total dust-storage capacity per drill is determined by the rate of dust production. For the Jackhamer, a volume of 1.5 cu. ft. is required for 4 hr. of operation. The storage space should be divided among the two or more stages of separation according to the proportion of dust collected in each.

Facilities must be provided for the safe removal and disposal of the collected dust without dangerous exposure of the attendant and others in the neighborhood.

The collector housing must be designed to withstand a negative pressure of 1½ lb. per square inch.

Source of Suction

The source of suction may be an electrically driven centrifugal compressor, compressed-air operated ejector, or other suitable apparatus. The ejector is convenient for use on isolated operations where electric power is not available and possesses the advantage of having no moving parts. Ejectors are not highly efficient, however, and are too expensive to use on large installations.

The ejector must be located downstream from the collector, since the air used to operate it must not pass through the filter. It is desirable also to place a fan in the same position so as to eliminate the wear due to the abrasive action of the dust.

Power Consumption.—The total pressure-loss through the exhaust system equals the sum of the losses through the hood, suction line and collector. This varies with the length of hose in use and the rate of air flow; in general, it does not exceed 1½ lb. per sq. in. Hence, with a flow of 80 cu. ft. per minute through each trap, the theoretical horsepower is approximately ½ hp. per drill.

Good Practice in Combatting Dust Hazards Associated with Mining Operations

By Donald E. Cummings*

(New York Meeting, February, 1935)

Certain dusts are dangerous when inhaled, but most hazardous of all dusts are quartz or other forms of pure crystalline silica. The inhalation of dusts containing silica in combination with other elements gives rise to changes in the lung which differ from those produced by pure silica. As a generality, silicates are less dangerous than pure crystalline silica and vary greatly among themselves in harmfulness.

Repeated analyses have shown that approximately 70 per cent of the total number of particles involved in any dust-producing operation are below 1 to 2 microns in diameter. (A micron is approximately 1/25,000 in.) Similar studies of dusts recovered from silicotic lungs post mortem have demonstrated essentially the same size frequency distribution, except that the upper limit of size retained in the air sacs is approximately 10 microns. Experiments carried on by Gardner and the author have further demonstrated that greater changes were produced by a fixed weight of very fine quartz particles (2 to 3 microns and smaller) than by the same weight of slightly larger particles (8 to 10 microns). Any attempt to control a dust hazard should therefore be directed toward the removal from the working atmosphere of particles less than 10 microns in size, and particularly those below 3 microns. All industrial hygienists agree on this matter, but there remains considerable dispute as to the lower limit of particle size capable of causing injury. It has been generally conceded, however, that particles of harmful dusts smaller than ¼ to ½ micron are not essentially concerned in the development of pulmonary fibrosis. This opinion is partly substantiated by the existence of large numbers of these small particles in practically all atmospheres that are not associated with any known hazard. Such particles are so small that they have very little mass, fail to settle even in quiet air, and probably are not easily retained in the lung. Until further research proves that they are involved in the dangers provoked by the inhalation of certain dusts, engineers will generally agree that it is better to disregard them.

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The determination of dust concentrations should be carried out by the method adopted as standard practice by the U.S. Public Health Service, and described in detail in reprint No. 1528 from the Public Health Reports. Since this method employs light field illumination at a magnification of 100 and reveals only those particles approximately 2 microns and greater in size, it appears desirable to supplement the standard technique by an additional enumeration made with dark field illumination. If a six-power objective and seventeen-power ocular are used with a Plankton condenser and a suitable source of artificial illumination, excellent dark field illumination can be secured which will visualize particles down to ¼ to ½ micron very readily. Determinations made with both light and dark field illumination provide valuable data on size frequency distribution well worth the additional effort required.

Mining operations that involve the disintegration and removal of ground are necessarily accompanied by dust. The instantaneous concentration at any working place depends upon the equilibrium established between the factors tending to produce dust and those influencing its elimination. The character of the ground, the nature and intensity of the operations, and the degree of confinement in which the operations are carried out govern the production rate, while the effectiveness of natural settling out, dilution, and special elimination methods determine the rate of removal.

In order to discuss good practice in alleviating hazardous dust concentrations produced by specific mining operations, consideration will first be given to the general conditions or procedures that mitigate the formation of excessive amounts of dust. Natural settling out of particles smaller than 10 microns in size plays a relatively insignificant role in the clarification of dusty mine air, as a calculation of the rates of sedimentation by Stokes law will readily disclose. Forced air movements and natural convection currents further retard the natural settling out process, which consequently is effective only in those working places that remain completely undisturbed for many hours or even days. Particles smaller than ¼ to ½ micron remain in suspension indefinitely, due to Brownian movement, and therefore are never removed by the effect of gravity. Expansion of dust-laden air tends to precipitate particulate matter, however, and this phenomenon can sometimes be utilized in mines by forcing air through restricted openings into large abandoned stopes or other old workings that communicate with surface.

Occasionally dusts are encountered that are hygroscopic, or which tend to form aggregates because they possess unlike electrostatic charges, and the settling out process is greatly accelerated by these phenomena. It is entirely possible that the application of dusts such as calcined gypsum might serve to augment these natural tendencies as beneficially as does rock dusting in coal mines in reducing dust explosions. Having secured the settlement of fine particles on to the solid surfaces surrounding a

dusty atmosphere, it is obviously essential that they adhere to those surfaces. Spraying with dilute solutions of materials such as molasses, glue or calcium chloride has been recommended, but the use of clean water is most satisfactory, and its application will be described in greater detail in a later section of this paper.

Ventilation

First among the most important general requirements for insuring safe working atmospheres in underground mines should be the provision of excellent general ventilation. In addition to supplying oxygen, this procedure dilutes and removes harmful dust concentrations, and the products of combustion of explosives, and aids in establishing proper temperatures and humidities. Normally clean surface air contains from 200,000 to 800,000 particles less than 10 microns in size per cubic foot when determined by the standard method adopted by the U.S. Public Health Service, though much higher concentrations are found near cities or large industrial establishments. Since concentrations of more than 5,000,000 particles of free silica dust per cubic foot of air are generally considered capable of producing harmful pulmonary changes, it is necessary to guard against contaminating the fresh-air stream before it reaches the dust-producing operations. Incoming air that has been polluted must obviously be supplied in greater volume to exert the proper diluting action at a hazardous working place, and good practice would suggest therefore that underground fresh-air currents should not contain more than 2,000,000 particles per cubic foot.

Experience suggests that for every man working underground approximately 300 cu. ft. of clean air per minute should enter the mine, and the cross-section of the smallest openings in the fresh-air stream should be such that the linear velocity will not greatly exceed 700 ft. per minute. Greater velocities tend to resuspend previously settled dust, add materially to the resistance, and exert too marked a cooling effect on the men. The source of air should be so located that the incoming current will not be contaminated by dust-producing operations on surface, and the stream should preferably be delivered underground through a separate and smooth-lined shaft, raise, or opening in order to minimize resistance and opportunities for the collection of dust that might later cause pollution. Obviously hoisting shafts should not be downcast, since spillage from the skips and headframe operations contaminate the incoming air, and counter currents and obstructions create considerable resistance. Mines can be ventilated to the best advantage only when two or more shafts are available. If only one shaft is provided, the cage road and ladder compartment should be tightly sealed from the rest of the shaft and used downcast while the skip compartment serves for the return of contaminated air.

The choice of forced or natural general ventilation will depend entirely upon existing conditions. If there is a sufficient year-round temperature differential between surface and underground, together with a suitable difference in elevation between the collars of the upcast and downcast shaft to force the required volume of air through the workings, then natural ventilation can be made satisfactory. When forced general ventilation is employed, fans should be selected that will provide the maximum contemplated volume requirements without overloading, and with sufficient head to exceed the greatest contemplated resistance.

The proper distribution of air underground demands careful thought and planning, so that clean air currents will be conveyed as directly as possible to the working places where they may dilute and remove harmful dust concentrations. Passages for these fresh-air streams should be as free from obstructions and sharp angular turns as possible, and should have a sufficient cross-section to keep the resistance relatively low, and the linear velocity within the previously stated limits. The smooth lining of airways, the construction of easements at sharp bends and the stream-lining of necessary obstructions not only reduce resistance but also reduce eddy currents that tend to resuspend previously settled dusts. When raises are used as a part of the distributing system they should be supplied with heavy wire-screen safety doors that will not offer dust-collecting surfaces nor unnecessarily obstruct the movement of air. General distribution should be regulated by the correct application of doors, regulators, stopping and brattice, so that appropriate amounts of air will be delivered to separate mining areas against the resistance offered by the complete passage leading to and from each of those areas. Abandoned or inactive workings should be completely sealed off from the fresh-air supply, while all doors and stoppages should be so constructed as to effectively prevent leakage.

Auxiliary ventilation will provide the final mechanism by which dusty operations will be partly controlled, but unless the general ventilating system is functioning properly, the use of auxiliary units is a waste of money and effort. Good practice in combatting dust hazards in underground mines dictates the installation of auxiliary ventilation in each rock heading where the working atmosphere contains more than 5,000,000 silica particles per cubic foot when all other protective measures are in operation. The objective of auxiliary ventilation should be to dilute the contaminated air around a dust-producing operation with a sufficient volume of cleaner air to render it innocuous, and further to direct the air movements so that clean air will be provided near the miner's face, while the dust-laden air is carried away.

Suitable electrically operated fans delivering air through galvanized pipe or canvas tubing are necessary adjuncts to any ventilation system for underground mines. Either high-pressure or low-pressure ejectors

are also useful, but do not compare with blowers for efficient operation. In general, fans should be located in special stations cut out quite well back in either the fresh or exhaust air passage, depending on whether forced or exhaust ventilation is used, and pipe or tubing led along the side of the drift or crosscut or along the back of the ladder compartment of a raise to within a few feet of the working face. The size of the fan and tubing will be determined, of course, by the number of men and the nature of the operations or conditions at the place of work. In selecting a small blower, consideration should be given not only to the volume required but also to the length of the tubing that eventually will be employed, since both low-pressure and high-pressure blowers are available and the low-pressure type is quite useless against the resistance offered by a long length of pipe.

Either forced or exhaust ventilation can be used satisfactorily, and sometimes a combination of the two will be desirable. If forced ventilation is employed, the tubing should be brought to within about 20 ft. of the breast and a return passage established by some means such as dividing the opening with a short piece of brattice. This feature is important in creating a sweeping movement across the face and preventing surging of the air in the heading. Tubing should always be elevated in forced ventilation so that the air tends to travel downward, carrying the dust produced at the face away from the breathing level. If exhaust ventilation is employed the pipe should be brought to within about 8 ft. of the breast and should remain near the floor so that the dust-laden air is directed away from the breathing zone. The efficiency of exhaust collection is materially increased by providing the intake with a "baffle plate" about as wide as the pipe diameter. This feature reduces slippage past the pipe and resulting development of eddy currents. It is quite practical to provide filters or water sprays to partially condition the air drawn through auxiliary fans if this is desirable—particularly when the units are of the high-pressure type. Tubing and piping must be maintained in good repair and kinks or sharp angular turns should be avoided by providing proper elbows. The discharge end of tubing should be fitted with a circular sleeve of metal to prevent fluttering, which adds to the resistance and produces an undesirable interrupted air flow.

Air that has been contaminated by dust-producing operations should be conducted to surface through a route that will expose the smallest group of workmen to its influence. Special ventilation raises, constructed in the footwall and connected by rock crosscuts with working levels or sublevels at one end, and with drifts on inactive levels at the other end, serve admirably for this purpose. Polluted air streams can then be converged at the hoisting shaft, which should be strongly upcast. In order to prevent contamination of fresh-air sources, ducts, crossovers and doors must be utilized as dictated by existing circumstances, since

it is exceedingly important to prevent recirculation, which cascades dust concentrations in a most amazing fashion. During development work it is sometimes impossible to bypass all of the contaminated air and part of it is bled into the fresh-air stream. Under these conditions the development work should be carried on after the regular shift has left the mine, or special procedures should be applied to condition the polluted air before it enters the fresh-air stream.

Use of Water

Second among the important general protective measures that may be utilized by all underground mines confronted with a dust hazard is the liberal and correct application of water. The reduction of dust concentrations by water depends upon the effective wetting of the particles so that they may be carried away in water or rendered more adhesive so that they will cling to solid surfaces rather than be expelled into the atmosphere; in addition small air-borne particles that would fail to settle when dry can be increased in mass sufficiently to permit them to settle promptly when properly wetted. Water for combatting dust hazards should be nearly neutral in reaction, free from sediment, or suspended particulate matter, and cooler than the underground air. Good practice here suggests that in so far as it is practical water should be applied to every operation surrounded by an atmosphere containing more than 5,000,000 silica particles per cubic foot of air when all other control measures are in use.

Water sprayed into a stream of unsaturated air evaporates rapidly, thereby tending to saturate and cool the air stream. If saturated air is sprayed with water of a lower temperature, the air is cooled and the excess moisture carried during saturation at the higher original temperature is condensed out until saturation is established at the lower temperature to which the air has been cooled. Since moisture condensed out of saturated air tends to form around existing nuclei of dust, the mass of small suspended particles is notably increased under these conditions, and their effective wetting and eventual settling assured. These principles can be effectively applied in most mines by the use of so-called "water curtains," which consist of a series of fine spray nozzles attached on 18-in. centers to a ½ or ¾-in. pipe extending up both sides and across the back of an air passage. A single curtain or a series of curtains can frequently be used to cool and dehumidify fresh-air currents that have become warmed by high surface temperatures or by passage through long timbered drifts or warm rock. More important, however, is the application of water curtains where contaminated air must come in contact with workmen before being vented to surface, as in long dead-end drifts. As a general rule, polluted air streams are warm and nearly saturated, and constitute

excellent vehicles for condensing out moisture on their load of suspended dust. Cool water sprayed into spent air streams by a series of curtains will efficiently remove dust until the temperature of the air has been lowered to nearly that of the water.

Water curtains also tend to accelerate the natural removal of dust from saturated air currents even where the temperature differential between air and water is not very great. This fact is accounted for by the purely mechanical wetting of suspended dust, and also by the fact that evaporation and condensation occur simultaneously in saturated air and consequently some condensation is occurring about dust particles under these conditions. The temperature, velocity, and humidity of the air as well as the temperature of the water dictate the number of sprays or the proper spacing of a series of curtains. The efficiency of such devices may be readily measured by determining the dust concentration of the air stream before and after passing through the sprays, bearing in mind that even well wetted particles will travel some distance before finally settling out. Under favorable conditions single curtains have frequently accomplished a 65 per cent reduction in the dust concentration of a polluted air stream. Three-way valves or extension arms should be installed to shut off the curtain when men or cars must pass.

Wherever underground fresh-air passages tend to dry out, water lines tapped at 100-ft. intervals should be provided so that the walls, floor and roof may be sprayed occasionally. Level stations at the fresh-air shaft should be habitually wetted down. Moist walls not only prevent dust from rising into the fresh-air stream, but also collect dust particles brought into contact with them by this stream.

Preventive Measures

Consideration can now be given to the application of preventive measures in reducing the hazard associated with certain mining operations. Blasting is the most potent of all the sources of dust underground. It is not unusual to find concentrations of several billion particles per cubic foot of air many minutes after a blast. Not only is a tremendous amount of dust generated by the fracture of the rock, but the concussion also sets into agitation dust particles that have settled out on surfaces several hundred feet adjacent to the blasting area. For this reason good practice dictates that all blasting should be delayed until the end of the shift, and that a considerable interval should elapse following the blast before a new shift enters the mine. It is difficult to adopt this procedure at all times, but whenever blasting expels extremely hazardous dusts into a fresh-air stream supplying other workmen, this operation should invariably be suspended until the end of the shift. The necessity for blasting in chutes or grizzlies can be overcome by the

provision of loading drifts, while boulders should be "popped" at the end of the shift. In order to minimize the dust hazard created by the concussion from blasting, the surfaces of a drift or the areas about the foot of a raise should be liberally sprayed with water prior to the blast. The dusts evolved in the blasting area can be confined by the use of water blasts, which are essentially cross connections between the water and air lines. These water blasts aid materially in dissolving the products of combustion of the explosive, and rapidly reduce the dust concentration to within reasonable limits. Dust counts should be taken at definite intervals after a blast in typical headings, in order to determine the minimum time that must be allowed before men may be permitted to re-enter the heading. The highest permissible concentration of silica dust that men may enter with safety when protected by respirators depends upon the efficiency of the respirator worn. The concentration of dust in the air breathed should not exceed 5,000,000 silica particles per cubic foot.

The extension of auxiliary ventilation should be the first operation performed after re-entering a recently blasted heading. Renewal of ventilation will hasten the clearing of the air, and if water curtains or special filters are provided they will assist in conditioning the contaminated fraction expelled. The immediate application of ventilation to such headings can only be undertaken when the surfaces have been thoroughly wetted down by water blasts. If the surfaces are not thoroughly wet, they should be sprayed, and all broken rock should be thoroughly soaked before mucking or scraping is started.

All loose rock should be kept thoroughly wetted down, and if scraping is common practice streams of water should be played on the muck piles during this operation. It is advisable to provide a depression in the floor filled with water, through which the scrapers must pass before reaching the scraper slide. Scraper hoist operators can frequently be placed behind water curtains, or provided with respirators, while hand muckers can be given neither of these protections.

Hard-rock drilling constitutes one of the chief hazards in mining, and good practice has long regarded wet drilling as essential. In South Africa only wet machines are permitted to be used, and all of these are required to feed water alone down the hollow drill steel, while in this country most wet machines feed both water and air down the drill steel. It is undoubtedly true that air bubbles expelled from the drill hole and bursting at the collar liberate unwetted dust particles into the atmosphere, and cause an increase in dust concentration. Because of the difficulties encountered in properly lubricating machines vented at the fronthead, wear is increased in the African-type machines, and if it is possible by other methods to attain safe working atmospheres during drilling they are to be preferred.

By far the greatest proportion of the dust produced in the drill hole is created during the collaring operation. For this reason good practice should include the provision of an auxiliary spray to be used during the collaring of all holes drilled in hard rock. This auxiliary spray may be a separate water line leading up to the breast, or it may be a short piece of hose teed into the regular water line on the drill and provided with a suitable nozzle. It may be held by the drill helper or it may be permanently attached to the drill. The face should be kept wet throughout the drilling operation, and all holes but the back holes should be drilled slightly down in order to preserve a water seal. Drill holes should habitually be blown with water rather than with air.

The exhaust from drills is another troublesome source of dust and oil mist. For this reason exhaust ports should be directed downward and backward, and drill manufacturers should be urged to provide ports that may be conveniently fitted with exhaust traps. The force of the exhaust air stream can be broken and the oil mist contained therein can be removed by the addition of a simple exhaust manifold, which aids materially in reducing the dust, fog, and noise attending the drilling operation.

Among the remaining mining operations, the air blowing of cars and chutes is one of the principal causes of dust. Cars should be scraped rather than blown, and the practice of air-blowing chutes should be minimized as far as possible.

Protective Equipment

Regardless of the perfection attained in the technique of combatting dust hazards by the methods described, there still remains a definite field of usefulness for special protective equipment. The most important of these devices is the ordinary respirator, and good practice should provide every man working in a hazardous underground atmosphere with the most appropriate form of respirator for the conditions involved. Sufficiently rugged respirators for underground use have not yet been manufactured, but simple filter-disk respirators can be obtained, which can be worn without great discomfort and offer considerable protection. In general, filter disks should be changed after about four hours use underground, because the moisture from both the exhaled and the inhaled air increases the resistance of the filter to such an extent that breathing is made difficult. This difficulty, which constitutes the chief objection workmen offer to respirators, may be overcome by providing extra filters.

Air-line respirators have a definite field of usefulness in mining, and can be used advantageously during drilling, particularly where there is a high concentration of dust created by many machines operating in a single heading or in raising. Air for these respirators is provided from the high-pressure lines after it has been filtered, expanded, heated, and

distributed by separate hoses to individual respirators. Workmen wearing these respirators are protected regardless of the condition of the atmosphere surrounding them.

Any method that provides for the removal of dust at its source will be more satisfactory than purely protective measures. Upon this principle is based the construction of traps designed to exhaust dust directly from the hole during drilling. These traps are being rapidly perfected, and should prove of great value in combatting the dust produced by drilling, particularly in raises where all other measures admittedly fail to accomplish the desired results. The installation of traps will not be a panacea for all the dust evolved in mining and should be regarded as an auxiliary means of protecting miners against the hazards accompanying the specific operation of drilling.

Inspection of Working Atmosphere

Finally, the key to the success of any program for combatting dust hazards lies in maintaining control over the effectiveness of the procedures adopted to reduce the dust. A properly trained engineer should be made responsible for constantly determining the dust concentrations throughout the working atmosphere of a mine, and for instituting the application of the methods outlined for reducing the amount of dust. His duties should include frequent inspection of all protective equipment and the preparation of records that will show the amount of dust present in each working place and the preventive measures that are in use. Experience has shown that the procedures described are effective in guarding against excessive concentrations of dangerous dust, and that when combined with medical control measures they will effectively combat a health hazard.

Methods of Handling the Silicosis Problem in Ontario

By G. C. Bateman*

(New York Meeting, February, 1937)

The Workmen's Compensation Act of Ontario was passed in 1915 and Miners' Phthisis was added to the list of compensable industrial diseases in 1916. Under this provision of the Act only about two cases were compensated per year.

In 1924, The Department of Public Health made a survey of miners in the four principal camps—Sudbury, Porcupine, Kirkland Lake and Cobalt—and the survey indicated that an alarming proportion of the men exposed to dust had silicosis in one or other of its several stages. A number of conferences were held between representatives of the Department of Mines, the Department of Public Health, the Compensation Board and the mine operators. The operators were advised that in the earliest stages of silicosis there was no physical disability and if men in that stage were removed from underground there would be little danger of permanent disability. They were further advised that if active steps were taken at once to compensate men that had some real disability and remove from the mines men in the early stages of silicosis, the first cost would be the greatest and thereafter their liability should decrease.

As a result of this advice, and having at that time no reason to doubt the findings of the survey, the operators agreed to do all that was asked of them and cooperate with the Government in the steps believed necessary to meet the situation, and to provide compensation for the men affected.

Amendments in 1926

Arrangements were made for the necessary amendments to the Mines Act and the Workmen's Compensation Act and these amendments were passed in 1926. The Amendment to the Mines Act provided that all underground employees must be the holders of certificates certifying that they were free from tuberculosis of the respiratory organs and further provided that the certificates must be renewed annually. The Government, however, at that time refused to adopt a form of certificate for applicants certifying as to their physical fitness for underground work. The Amendments to the Compensation Act defined both silicosis and

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tuberculosis and adopted the South African basis of classification for silicosis in its three stages of anteprimary, primary and secondary. These are practically identical with the first, second and third stages used in the United States.

The Amendments further provided that to be entitled to compensation a man must have been working at the date of or subsequent to the passing of the amendments, and that he must have had five years proven exposure in Ontario mines. Exposure prior to the passing of the amendments was allowed provided the claimant was working at the time of or subsequent to the passing of the Amendments.

At that time it was considered advisable, both for the companies and for the employees, that men with silicosis should be taken out of the mines. To ensure this the Compensation Board in advising a man that he had been classified as a silicotic notified him that if he continued the work in which he was exposed to silica dust he would not be entitled to any further compensation, whether or not his disease progressed.

The award made for anteprimary or first-stage silicosis was $500. Because there was believed to be no physical disability with that stage of silicosis, the award was not considered as compensation but as a rehabilitation allowance to carry the man over the period of establishing himself in some other occupation. The award for primary or second-stage silicosis, which implied some physical disability, was $1000, and for secondary or complete disability the regular award for total disability, which is two-thirds of average earnings with a maximum of two-thirds of $2000 per year.

An examining station was established at Porcupine in charge of a doctor who was believed to be expert in diagnosing silicosis. At first this station was operated by the mines themselves but subsequently, at the request of the operators, was taken over by the Workmen's Compensation Board. The results of the early examinations showed so high an incidence of silicosis that questions naturally arose as to the accuracy of the diagnosis. It was considered advisable to get advice from South Africa, where the technique of examinations had been well established, and Dr. J. M. Smith, Chairman of the Silicosis Referee Board in South Africa, was retained to make a survey of the whole situation.

After spending three months in checking the work done up to that time, Dr. Smith reported that the actual number of silicotics was very much smaller than had been stated, and that many men having only a general fibrosis had been classified as silicotics. Dr. Smith's work resulted in the adoption of the South African standards for diagnosis in Ontario, but in the meantime a large present and prospective liability had been created for men who were not entitled to it.

As stated before, the mines had been advised that the $500 payment to men with anteprimary silicosis was for rehabilitation rather than compensation, and that with this payment to them, the liability of the mines would be practically at an end. It developed, however, that the legal position was different. Any award made by the Board, including the $500 to nondisabled silicotics, could be considered only as compensation, and compensation could be paid only for disability. It followed therefore that any increase in disability or increase in progression of the disease was considered as the result of the original disability, and for this the mines were held responsible.

A workman was considered as having secondary silicosis: (1) when there was serious and permanent impairment of working capacity from uncomplicated silicosis, (2) when tuberculosis was present with any compensable degree of silicosis. Experience showed that silicosis did progress, therefore, although this was not apparent for some years, every anteprimary silicotic was considered as a potential total disability case, with an average estimated cost of $11,000. Eventually, when an award of $500 was made to an anteprimary, an additional progression liability of $10,500 was set up and the industry was asked to provide this sum as well.

As time went on, it was also found that removing the men from underground did not necessarily improve their condition or arrest the disease. In fact, as a general rule removal accelerated progression. While this was contrary to the general view, we were forced to the conclusion that there was little advantage in removing a known silicotic from his ordinary work, as a careful analysis of progression rates indicates little if any difference in favor of those removed. This, of course, is subject to the qualifications that every effort should be made to reduce the dust hazard, that silicotic workmen should not be allowed to work where the concentration of dust is high, that where periodic examinations show a rapid development of fibrosis the workman should be removed from dust exposure before he reaches the silicotic stage, that where there is a definite suspicion of tuberculosis the workman should not be allowed to work in dust exposure, and that no workman with pulmonary tuberculosis should be allowed in any position where he will be a menace to his fellow employees.

Of nine men in one mine classified as silicotic in 1926, eight are alive today, the ninth having died of pneumonia. Of the eight, aged now about 52, two were removed to the surface and have been employed there since and the other six have been continuously engaged in underground work. Except in one man, who is 61 years old, there is no evidence of slowing up. One of these men, who was rated as a primary or second stage silicotic in 1926, today, at 54 years of age, is active and healthy

looking. It is, however, only fair to say that the basis of classification has been substantially raised since 1926.

Theoretically the removal of silicotics from dust exposure should retard the progress of the disease. In actual practice our experience has been different. As a general rule, the occupations that involve dust exposure are the highest paid in industry in Ontario. A man leaving one of these positions might have to take lower pay. He might, on account of industrial conditions, be unable to get any job. Miners as a class are difficult to rehabilitate. Removal from the mine meant that their economic status was probably lowered. The award was soon spent and with the money gone and no job they often lived under conditions that accelerated progression. In addition, and perhaps most important of all, the psychological effect of the notification from the Board was very bad. Many of the men practically gave up hope and made little effort to help themselves. There was a lowering of morale, with a consequent bad effect upon a man's physical condition. These factors offset the benefits that might have been expected from removal of men from underground, and even if the economic status had remained the same it is doubtful whether removal would appreciably retard the rate of progression.

Amendments in 1933

Tuberculosis itself is not a compensable disease, but tuberculosis with any stage of silicosis is considered as total disability. Silicosis predisposes a man to tuberculosis and, as the Ontario Act was interpreted, the development of tuberculosis in a silicotic was considered as the result of the silicosis and as such was compensable. It did not make any difference under what conditions, or how many years after a silicotic had left the industry, if tuberculosis developed the industry was held liable. In effect the industry was required to provide insurance against the development of tuberculosis in a silicotic as long as the man lived.

These conditions created what the industry believed to be not only an onerous but an unfair burden. A large liability had been created in respect of cases which in the light of later information should not have been classified as silicotic. Men that would have been happier, and probably healthier, working were being removed from the mines. The industry felt that there should be some reasonable limit of time within which the mines would be liable for additional compensation as the result of the development of tuberculosis after a man left the employment in which he was exposed to dust.

Neither the men nor the mines were satisfied, therefore further conferences were held between the mines, the Department of Health, the Department of Mines and the Compensation Board, with a view to amending the Act. The Amendments were introduced in 1933. Briefly, they provided that the old classification in three stages should be done

away with—and with it, of course, went the lump-sum awards for anteprimary and primary cases. While not provided for in the Act, it was also agreed that men in the early stages of silicosis should be allowed to continue in their ordinary occupation—provided, of course, there was no tuberculosis, or even a reasonable suspicion of tuberculosis. The Amendments further provided that while the industry would be responsible at any time for increased disability as the result of the progression of uncomplicated silicosis, it would not be responsible for increased disability as the result of tuberculosis unless the tuberculosis developed within two years from the time of leaving employment in which the workman was exposed to dangerous dust.

The draftsmanship of the Amendments was faulty, so that there is some question of the interpretation. As a result, individual cases are very largely dealt with on their merits, instead of in a strict legal manner. While this has certain disadvantages, it also has something to commend it, particularly when there is, as in Ontario, a competent, intelligent and impartial Board. There is also some question as to whether or not two years is too short a period, and this may possibly be lengthened.

Competent Diagnosis

The principle upon which practically all compensation acts are based is payment for impairment of working capacity. In connection with silicosis, this is difficult to determine. Silicosis is an insidious disease which among miners in Ontario takes usually 15 to 18 years to develop. The diagnosis, particularly in the earlier stages, is dependent almost entirely upon an X-ray film. There may be few if any physical or clinical signs. It is quite probable that a man with anteprimary silicosis could be examined by a clinic of doctors and that without the X-ray film they would be unable to say that there was anything wrong with him.

A good deal of what may be read into a film depends upon the technique of taking the film. Its interpretation is to a considerable degree a matter of personal opinion. Other diseases and chest conditions may give in the film an appearance so closely simulating silicosis that differentiation is practically impossible. There are few normal adult chests. The great majority of X-ray pictures of adults that have had no industrial dust exposure show some scar tissue and general fibrosis. Often there is a considerable development of general fibrosis, and some cases have been found of people that have had no industrial dust exposure that if they had been miners would certainly have been diagnosed as silicotic.

We believe there is a tendency on the part of medical men to estimate disability on the basis of a pathological condition as indicated by an X-ray film rather than on actual impairment of working capacity. In any event, doctors are not always the best judges of what constitutes working disability, and the X-ray film alone is not a satisfactory basis for

estimating disability for compensation purposes. We also believe that once a man has been classified as silicotic there is a tendency to claim that any increase in disability is attributable to the silicosis, although it may be the result of advancing years or due to causes that cannot be definitely laid to silicosis. This is particularly true when the heart is involved, although there is a tendency toward heart conditions in older miners that have no silicosis. However, because the heart condition *might* be the result of silicosis, it is generally attributed to it.

I should like to emphasize the necessity of having competent examiners, who have been specially trained in the diagnosis of silicosis by both clinical and X-ray examinations. The opinion of the general practitioner, and even that of the roentgenologist and expert chest man, may be of little value. The matter is so important that the necessity for competent medical advice cannot be emphasized too strongly. We consider ourselves fortunate in Ontario in having in charge of this work doctors that are both competent and sincere. We may not always agree with them, but this does not prevent us from acknowledging their ability.

Administration of the Act

Administration naturally falls into four divisions—examinations, compensation, assessments and costs.

Examinations.—The Ontario Act provides that all underground men shall be the holders of certificates, and that they shall be examined annually. There are three permanent examining stations, and a traveling examiner for the outside mines. Each of these clinics is in charge of a doctor that has had special training and experience. These doctors do the routine examinations and issue or refuse the certificates. The doctors and the stations are under the jurisdiction of the Compensation Board. In addition, there is a Referee Board, consisting of doctors of the Provincial Department of Public Health, and no claim for silicosis is allowed by the Compensation Board until the claimant has been examined by the Referee Board. The permanent stations are located in the principal camps, and can examine an average of 4000 men a year each.

The doctors also examine all applicants. If silicosis is to be controlled, the standard of men allowed to enter the industry must be high. Standards for acceptance have been adopted and have gradually been raised, and the initial examinations have been made more rigid in order to keep out of the industry men with the type of chest that would indicate their susceptibility to tuberculosis or silicosis.

Uncomplicated silicosis is not a serious problem. The dangerous factor is tuberculosis, which is the most active known agent in the development of fibrosis of the lungs. A tubercular person may be just as dangerous on the surface as underground, and arrangements are now being

made to have the examination of surface men made a regular part of the work. Men with tuberculosis are not allowed to work underground and the doctors remove men from underground work when there is a definite suspicion of tuberculosis.

Compensation.—A claim for silicosis compensation may be made by the claimant himself, by his employer, by the doctor, or by anyone acting on his behalf. The first thing the Board does is to obtain full particulars from the claimant and his employer, including a statement of his earnings, and to establish whether or not he has had five years proven exposure in Ontario mines. If the latter cannot be proved, the claim is not allowed. If the man is eligible for compensation, the claim is passed on to the Referee Board, which examines the claimant and reports to the Compensation Board. When there is tuberculosis with silicosis the man is classified as a total disability. In other cases the Referee Board may estimate a percentage of disability due to silicosis, or may find that the man does not have a compensable degree of silicosis, or may find that the man does not have silicosis within the meaning of the Act. When the Referee Board has made its report, the claims officer and the rating officer complete the file and submit it with their recommendation to the three members of the Compensation Board. Except in cases of total disability from tuberculosis, all claims for silicosis are generally passed upon by the full Board, which may or may not, but usually does, accept the opinion of the Referee Board as to the degree of disability, if any.

Except in the case of silicosis complicated by tuberculosis, total disability compensation amounts to two-thirds of average earnings up to a maximum of two-thirds of $2,000 a year, with a minimum of $12.50 per week. Partial disability is paid on a proportionate basis. Tuberculosis is not a compensable disease, and because all men are liable to it, and because a certain proportion of the men classified as silicotic would develop tuberculosis whether or not they had silicosis, the rate of compensation for silicosis complicated with tuberculosis is 50 per cent of average earnings, up to a maximum of 50 per cent of $2000 a year.

Medical aid, including hospital or sanatorium costs, is payable in addition to compensation and there is no limit to the amount of medical aid that may be paid.

If death ensues, compensation is also payable to dependents. A widow receives $40 a month for life, or until remarriage, with an additional $10 a month for each child up to the age of 16 years. The pension to the widow and children is the same no matter what compensation the claimant may have been receiving; provided, of course, that death is considered to be the result of the disability.

The total average estimated cost of a total disability silicosis case is approximately $11,000, divided roughly as follows: compensation to

claimant, \$3000; medical and sanatorium treatment, \$4000; pension to dependents, \$4000.

Silicosis Assessments.—The original survey of the Department of Health indicated a considerable variation in the incidence of silicosis as between the different mining camps of Ontario, and this difference has been confirmed by subsequent periodic examinations.

There was considerable objection to having silicosis made a part of the accident rate, and under an arrangement between the Ontario Mining Association and the Compensation Board the following basis was agreed upon.

For the purpose of silicosis assessments only, the mines of the Province were divided into five groups: (1) Porcupine, (2) Kirkland Lake, (3) Sudbury, (4) silver mines, (5) all other mines. If a particular camp included in group 5 grows to a sufficient size, a new group may be established.

Each group bears the cost of the silicosis charged against the mines of that group. For purposes of record, a silicosis case is charged against the mine where the man last worked, even although the man may have worked in other mines of the same group, or in mines of other groups. A good deal of consideration was given to this decision, but it was believed impossible to allocate cost according to exposure. The development of the fibrosis is not necessarily proportionate to the exposure in any one mine. Many of the mines in which claimants claimed exposure were closed down before the Act came into effect, and in any event the Compensation Board is interested only in proving five years exposure. While for purposes of record the case is charged against the mine where the claimant last worked, that mine is not individually liable for the cost.

Each year the Board estimates the cost for each group, and the amount is divided among the members of that group according to dust shifts. In estimating cost, the Board not only makes provision for the immediate cost of each case allowed but also includes a reserve for progression. The dust shifts are the shifts worked by all underground men, by men employed in dry-crusher stations on the surface, and by men employed in the grinding department of assay offices. Millmen engaged in a wet process are not included.

Each year the individual mines are required to report the dust shifts worked. The Board having determined the total dust shifts for a group, and having estimated the cost of silicosis for that group, divides one by the other to determine the cost per dust shift. The cost for each mine is then found by multiplying the dust shifts of the individual mines by the average cost per dust shift for the group. To this must be added the cost of examinations, Referee Board, etc. The total cost of these items is divided by the total dust shifts reported from all mines in the Province,

and a flat rate per dust shift is obtained, which is added to the cost per dust shift of the individual mines.

While the cost per dust shift is the same for each mine of a group, it may and in fact does vary considerably for the different groups. For example, the cost per dust shift for the 1936 silicosis assessment was as follows: (1) Cobalt, $0.0933; (2) Kirkland Lake, $0.0361; (3) Porcupine, $0.0919; (4) Sudbury, $0.0194; (5) all others, $0.0896. These figures are considerably below the general average and are given only to show the variation among the camps.

Costs.—There is probably no other industrial disease about which more loose talking is done, therefore some actual figures based upon experience in Ontario may be of interest (Table 1). In considering these figures it should be remembered that the provisions of the Ontario Workmen's Compensation Act are more liberal than those of most such acts, and that the 1926 and 1927 examinations included an accumulation of cases prior to 1926. Many men in those years were classified as silicotic that would not be so classified today, but these cases did create a large liability, both direct and potential. Also, as a general rule, the workman receives the benefit of the doubt.

The silicosis amendments were passed in 1926. It is therefore a fair assumption that in taking the figures from 1926 to 1936 inclusive, and including miners' phthisis with silicosis, both the number of cases and the cost are greater than they should be.

For purposes of comparison some information is also given regarding accident compensation rates, and deaths from accidents underground, as the latter provide a fair basis of comparison for men subject to the dust hazard.

General.—The figures in Table 1 show that from the point of view of both cost and hazard silicosis is not nearly as dangerous as ordinary underground work. Every effort is being made however to improve conditions and reduce the hazard. That these efforts are meeting with success is shown by the reduction in the number of new cases. The average number of cases allowed by the Board from 1926 to 1936 inclusive, was 36, while for the past five years the number has been reduced to 18. The period of exposure for the development of silicosis has also been lengthened from an average of between 10 and 11 years to an average of approximately 18 years.

This improvement has been brought about as a result of the active and intelligent cooperation of the industry. Recognizing that if silicosis is caused by dust, the removal of the dust will remove the cause, large sums have been spent by individual companies in ventilation. As the dust that causes the damage is practically invisible, dust sampling is used to give definite information regarding conditions. A great deal of atten-

tion has been paid to blasting practice, as blasting is believed to be the chief cause of dangerous dust in mines. Water sprays after blasting and at dusty chutes are extensively used. Considerable work has been done with dustless fronthead drills, and with respirators to be worn in dusty places. Some experimental work has also been carried on with various kinds of tamping to determine the silica content of clay tampings and with a view to finding a satisfactory substitute for clay tamping.

TABLE 1.—*Data on Silicosis in Ontario*

Period of Time		Average
1926–1936, incl.	Number of men employed per year in the mining groups	14,000
1926–1936, incl.	Number of men employed underground per year.......	8,500[a]
1926–1936, incl.	Number of silicosis and miners' phthisis cases allowed per year by the Workmen's Compensation Board......	36
1926–1936, incl.	Number of deaths per year allowed by the Workmen's Compensation Board as deaths from silicosis..........	9.3
1926–1936, incl.	Number of cases per year terminated otherwise........	3.4[b]
1926–1936, incl.	Number of deaths per year from accidents to underground employees only............................	32
1932–1936, incl.	Number of men employed underground per year for the five years..	10,000[a]
1932–1936, incl.	Number of silicosis cases allowed per year by the Workmen's Compensation Board........................	18
1916–1936, incl.	Accident assessment rate per $100 of payroll for all industries in Ontario coming under the Workmen's Compensation Board..................................	$1.17
1926–1936, incl.	Accident assessment rate per $100 of total payroll for mining groups only..............................	$2.63
1926–1936, incl.	Silicosis assessment rate per $100 of total payroll for mining groups only..............................	$1.17[c]

[a] Estimated.

[b] The average of 3.4 cases per year which terminated otherwise than as deaths from silicosis includes silicotics that have died from accidents or other causes and also silicotics that have been repatriated and regarding whom no further information is available.

[c] While silicosis assessments are levied only on dust shifts, the cost for purposes of comparison is calculated at the rate per $100 of total mining payroll, as it would be difficult to separate out the accident rate per $100 of payroll for those men exposed to dust.

Through the cooperation of the Department of Health much more attention is being paid to the control of tuberculosis among the general population of mining camps. The opinion seems to be gaining ground that the development of tuberculosis in miners is due largely to outside contacts. Some of the mines also have their own nursing service to follow up on tuberculosis contacts.

Research

In addition to measures for the control of silicosis adopted by the individual mines, the industry through the Ontario Mining Association finances both medical and engineering research.

Medical and pathological research is carried on by the Banting Institute under the direction of Sir Frederick Banting, and in cooperation with the Technical Silicosis Research Committee of the Association. During the past two years the Association has contributed $30,000 to the Banting Institute for this work.

The Silicosis Research Committee was appointed by the members of the Ontario Mining Association. It consists of four technically trained men, employees of mines in the different areas, who have made a special study of this subject and who have the research type of mind. The Committee employs a full-time engineer whose work consists largely of various phases of engineering research, including detailed studies of ventilation at different mines, on which reports go to the individual mines and to the Mining Association; studies of underground conditions and measures for dust control at the different mines; determination of the pH of mine atmospheres; the collection of underground dust, ore and rock samples for experimental purposes; the collection of data from various mines in connection with the Banting Institute research; and the collection of data regarding experimental work and new developments both in Ontario and other places. The Association keeps its members advised of the work of the Committee.

The intelligent approach to the problem and the active interest of the industry has had a marked effect in reducing both the incidence and cost. Silicosis is essentially an employer's problem, and the sooner this fact is recognized by an employer the better off he will be. The work done to date indicates that silicosis can be controlled. It is quite within the realm of possibility that measures for control can so lengthen the period of exposure for the development of silicosis that its onset will be coincident with the age when a man would ordinarily cease gainful employment. If this occurs, the burden on industry need not be heavy.

DISCUSSION

(*John J. Forbes presiding*)

D. Harrington,* Washington, D. C., asked to what extent the so-called dustless head was used in drilling, and what were the effects of this head on the speed and ost of drilling, to which Mr. Bateman replied that it was his recollection that it gave dust counts from ¼ to ⅓ the normal count for wet Leyner drills. The cost of maintenance and the speed of drilling are approximately the same as for the older type.

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* Chief Engineer, Safety Division, U. S. Bureau of Mines.

R. E. Larry,* Pittsburgh, Pa., asked if they could drill as deep as formerly, say 8, 12 or 16 ft., and Mr. Bateman replied that they are doing so.

In response to a question from B. F. Tillson,† Upper Montclair, N. J., G. H. C. Norman,‡ Copper Cliff, Ont., said that the dustless head originated in South Africa, the principle being to keep air out of the hollow drill steel.

H. J. Mutz,§ Copper Cliff, Ont., added that ports in the cylinder by-pass the air so that it does not blow out through the drill steel. Only water is forced through the steel. Exhaust ports are turned away from the fan.

D. Harrington asked about the difference in the speed of drilling, and stated that he has reports of tests made using dustless heads, which showed the speed to be decreased 25 per cent.

H. J. Mutz informed him that they originally suffered a decrease in drilling speed of 11 to 15 per cent. They were using CP6 and Waugh 7K11 drills, averaging 6 in. per min. They are now averaging 9 in. per min. with drills of the same bore and stroke, the drills having been so improved.

G. S. Rice,|| Washington, D. C., pointed out that it is not a matter of taking dust out but, by wetting, preventing a dust cloud with the air, which passes through the drill steel.

B. F. Tillson asked if it were true that the addition of the head reduced the capacity of tappet drills 10 per cent.

H. J. Mutz said that there is a loss of efficiency in the tappet drill, due to the absence of air cushioning, but that it was not more than 10 per cent.

G. C. Bateman added that there is some loss in up holes but nothing like 10 per cent in down holes.

D. Harrington asked about the comparative amounts of dust produced by dull and sharp bits, but Mr. Norman had no data on this point. Mr. Harrington added that this seems to be a difficult thing to establish, a British publication reporting much more dust with a dull bit than with a sharp one. The Bureau of Mines has found the reverse to be true in wet-drilling quartzite.

B. F. Tillson commented that this would be entirely a matter of the composition and structure of the rock.

In reply to Mr. Larry, Mr. Mutz said that the drilling of the vertical stoper holes is similar to drilling horizontal or down holes. The men are provided with rubber clothing and do not object to the drainage from the up holes. However, more dust is produced from overhead vertical holes, particularly in headings. Mr. Norman amplified this with the explanation that the concentration of dust also depends largely on the circulation of air in the heading which in the case of raises is mainly maintained by the exhaust from the drill. On drilling up the water gets away too freely to properly prevent the formation of dust with the net result that up holes in raises now give the worst dust condition from drilling.

In response to a question from Mr. Harrington, Mr. Norman said that he would not want to imply that all modern respirators are very effective. Those with the

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† Consulting Engineer.

‡ Test and Ventilation Engineer, International Nickel Co. of Canada, Ltd.

§ International Nickel Co.

|| Chief Mining Engineer, U. S. Bureau of Mines.

largest filter area are usually the most effective. The efficiency of many respirators on the market is measured not by weight but by dust count under actual working conditions with a konimeter and dark-field counting is less than 60 per cent and sometimes as low as 35 per cent for those of restricted filter area.

He referred to a respirator in his paper[1] that was developed by Hollinger Gold Mines and the design given to different respirator manufacturers to make up. It has large inspiration and expiration valves in the face piece and a large flexible rubber tube to the filter bag worn on the chest. The filter bag is 3-ply of surgical cotton between two layers of felt about ten inches square, protected with a covering of duck with eyelets to let in air. The air passage is so free it does not cause fatigue to the wearer. It is used regularly by blasting crews when necessary to return to blast the square-up in development ends after blasting the cut; and in the tests made under actual working conditions with the konimeter using dark-field counting, passed less than 300 particles per cubic centimeter from the densest cloud of blasting smoke.

D. Harrington called attention to the great difficulty in getting men to wear respirators and asked if the men wear them while drilling and mucking. Mr. Bateman said that they are regularly worn only at crusher stations, and Mr. Norman added that they may use the new type of respirator in drilling raises.

B. F. Tillson asked as to the lower limit in size for harmful particles, citing recent opinion to the effect that the more minute particles are not retained in the lungs, and hence are harmless. A. J. Lanza,* New York, N. Y., was doubtful but said that some research workers hold to the view that particles below ½ micron do not have much, if any, effect.

R. E. Larry asked about the relation of compensation for silicosis to the statute of limitations, and Mr. Bateman reported this matter to be in a state of flux, but there is no limit of liability for uncomplicated silicosis. However, when silicosis is complicated with tuberculosis compensation is not paid unless the tuberculosis develops within three years of the cessation of employment in an occupation where the workman was exposed to silica dust.

Mr. Bateman said that in Ontario substantial amounts are being paid on partial ratings. He also commented that a doctor may not be the best person to establish such ratings, inasmuch as a doctor has no knowledge of a miner's day's work. Furthermore, when a man appears with silicosis and other chest conditions, say a heart disorder, the doctor tends to attribute the disorder to silicosis, because he cannot say for certain that that is not the cause.

A. J. Lanza felt that physical disability might be due to uncomplicated silicosis, and that it might be necessary to provide compensation therefor, as well as for silicosis with tuberculosis. He asked if any appreciable amount of uncomplicated silicosis had been found and Mr. Bateman could recall but one case of death from uncomplicated silicosis. In many cases partial disability has been paid for uncomplicated silicosis.

As to percentage of quartz, Mr. Bateman said that in compensation matters, the critical thing is the amount, not the character, of the dust. Both silicosis and pneumonoconiosis are compensable and the real concern is with how much dust the workman encounters, not how much free silica it contains.

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[1] G. H. C. Norman: Methods of Sampling and Dust Determination in the Mines of Ontario. A.I.M.E. Tech. Pub. No. 857 (*Min. Tech.*, Dec. 1937).

* Assistant Medical Director, Metropolitan Life Insurance Co.

D. Harrington said that he had recently been confronted with a new problem which came from a coal mine, but might equally well appear in a metal mine. That is the effect of stirring up pulverized locomotive sand on grades, or starting points, where locomotives sand the rails. The trip rider and motormen are exposed to this siliceous dust, which is causing some concern.

G. C. Bateman said the question has not risen with them. Their mine atmospheres are clear except after blasting, which is the real dust raiser, or near loading chutes and underground crushers. Mr. Norman related that samples taken in the main haulageways, which are on fresh air, showed little more dust than the atmospheric samples.

D. Harrington felt that this question has a corollary in the rock dust stirred into the air by haulage. He has always felt that this presents a negligible hazard to health, but if dust from the rail sand is a hazard, may not the rock dust be one too?

B. F. Tillson asked what work had been done on the pH of dust in aqueous suspension, stating that it might vary from acid to alkaline. Mr. Bateman said that much work has been done on this. The cilia of our respiratory passages are constantly in vibration to remove dust particles. Microscopic examination has shown that an acid environment retards this movement. This is also true of an excessively alkaline environment, but with moderate alkalinity the activity of the cilia is normal or slightly accelerated.

Poorly ventilated mine atmospheres are acid. Their mine with lowest silicosis showed slightly alkaline atmosphere. However, Mr. Norman warned that even a heavy dust, when fresh, may be alkaline, and the pH is not always a measure of atmospheric staleness.

W. T. Schaller,* Washington, D. C., mentioned the work of R. C. Emmons who concluded that, since the lymphatic lung liquids are naturally alkaline they will dissolve small quartz particles, depositing nodules of silica elsewhere, with resultant silicosis. To stop this he would add more dust which must carry electrical charges opposite in sign to those of the silica. Iron oxide is suggested for the purpose.

Mr. Norman hardly felt this to be an engineering approach to this matter, as they have found that increasing the quartz or other dust does not always work out as expected. Mr. Harrington pointed out that, on that theory, our iron mines should be free from silicosis, whereas they have much of it.

L. P. Wood,† New York, N. Y., asked for a summary of the methods of keeping dust down in tunnel work, as his Board has 85 miles of tunnel to drive. Mr. Harrington assured him that the two chief weapons against dust are ventilation and water. Always have the tunnel face well ventilated. Use wet drilling, and spray before and after blasting. Wet the tops of loaded cars. Do not blast during the shift, if it is possible to avoid it. If it is necessary to blast with men in the tunnel, do not let them return to the face until the dust has been removed from the air by ventilation and water sprays. If they must go back sooner, use respirators as a last resort. The occupational dust hazard may become such an acute problem that tunnels and metal-mine passageways will be driven in pairs with interconnecting crosscuts to facilitate ventilation, as has long been the case in coal mining.

H. J. Mutz described the driving of a long heading in the International Nickel mine. Ventilation was carried to the face by means of piping, provision being made

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* Mineralogical Chemist, U. S. Geological Survey.

† Board of Water Supply of the City of New York.

for exhausting or blowing the air. Water sprays were also kept at the face, the water and air hoses being interconnected so they could be turned on a considerable distance from the face. When miners must return to face as soon as possible, it was found better to exhaust the air.

Dust tests showed the air to be clear 30 min. after blasting. The intake to the auxiliary ventilation equipment was in fresh air.

G. H. C. Norman stated that the dust set up by a blast can not be knocked down with a spray. It may be held down by wetting beforehand, but once formed after the blast, it can only be removed by ventilation.

C. S. Hurter,* Wilmington, Del.—In regard to wetting the muck piles to reduce the amount of dust, this should go well beyond the bare moistening stage. Moist muck holds the fumes after blasting, resulting in more trouble during its removal, than with dry muck. Therefore the muck pile should be thoroughly drenched so as to wash the fumes out of the blasted material. In the days before the widespread use of the compressed-air drill and hand drilling was common, a regular fire hose was used in some mines, where long drifts were being driven and the ventilation poor, to play on the sides of the drift and to wash the fumes from out of the muck pile. This not only reduced the dust hazard to a minimum but gave the returning men better working conditions.

H. P. Greenwald,† Pittsburgh, Pa., asked R. D. Hall,‡ New York, N. Y., if the Safety in Mines Research Board adds any chemicals to the drilling water to increase its surface tension, thereby making it wet the dust better.

E. D. Gardner,§ Tucson, Ariz., reported that soap suds had been used experimentally in the drill water at a mine at Fresnillo, Mexico, but he did not know whether it did any good.

G. C. Bateman and G. H. C. Norman said that attempts had been made to use soap, castor oil and water glass, with inconclusive results.

D. Harrington reported only a little interest to date in the United States in wetting agents, although some cooperative work has been done with explosives manufacturers. Foaming agents are regarded favorably in Great Britain. He felt it necessary to regard dust as a hazard, regardless of its composition, citing two cases of silicosis in talc mills, which led to disability in one case and death in the other. Mr. Tillson added that any dust produces irritation which may lead to disability.

G. S. Rice stated that the Public Health Service had reported that certain silicate dusts from Washington State as indicated by samples would be dangerous to use as rock-dusting materials in coal mines, because they are likely to cause fibrosis.

A. J. Lanza stated that the harmful effects of other dusts is in doubt. After many physical and X-ray examinations of workmen here and abroad, who had been exposed to dusts from feldspar, gypsum and silica, and after much experimentation with animals in which the animals lived in dust-bearing atmospheres for years, and had dust preparations injected into the peritoneal cavity, so far nobody has shown that the typical reaction known as silicosis can be produced by any dust except free silica.

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* Wilmington Trust Co.

† Supervising Engineer, Pittsburgh Experiment Station, U. S. Bureau of Mines.

‡ Engineering Editor, *Coal Age.*

§ Supervising Engineer, Southwest Experiment Station, U. S. Bureau of Mines.

The effects of asbestos dust are entirely distinct from those of silica. Peribronchial thickening (fibrosis) results largely from dusts other than silica.

R. E. Larry agreed that we can not have silicosis without silica.

C. F. Jackson,* Washington, D. C., explained Mr. Littlefield's report of getting less dust with dull than with sharp bits in wet drilling, by pointing out that the input of water, per inch drilled, was much greater with dull bits because more time is required to drill an inch, whereas the rate of flow of water through the steel and into the hole is constant.

J. B. Littlefield,† Pittsburgh, Pa., stated that the Mt. Weather station of the U. S. Bureau of Mines is provided with X-ray and spectroscopic equipment with which it is hoped to identify extremely fine material particles.

G. H. C. Norman added that workers at Banting Institute are able to determine the quartz content of a dust within 10 per cent, by X-ray analysis, and that they soon hope to get within 5 per cent.

E. D. Gardner reported that a mine in northern Mexico had subjected its men to physical examination and found practically all men who had worked underground over five years to have more than the permissible 10 per cent involvement. The mine has used wet drilling for the past 10 years. The examination was made by the company doctor. It was also found that a group of cowpunchers also showed more than 10 per cent involvement.

L. P. Wood asked if there were any difference in the effects of high and low-temperature quartz, but Mr. Norman said that quartz from all over Ontario had been tested, and no differences in toxicity had been found.

With respect to the Mexican experience, Mr. Bateman emphasized the necessity of having a competent medical specialist make the examination, as a general practitioner is not dependable in this respect.

O. M. Schaus,‡ Montreal, Wis., stated that his company was most keenly interested in the safety and health of their employes, and had spent and are still spending a great deal of money to eliminate the hazards to safety and health. Organized controlled ventilation is the prime factor and to be thorough in establishing and maintaining proper ventilation, it is highly essential to have a ventilation engineer as part of the organization. As much as possible, dust must be prevented from forming. Once it has formed, protection must be given the men through adequate ventilation and the use of respirators. The establishing of adequate ventilation must necessarily be an ever-changing and improving process. Respirators of the approved Bureau of Mines type should be used even though it is felt that ventilation is quite satisfactory so that the men will become dust-conscious and, of course, to obtain the value of reducing the dust hazard by the use of the respirator. We have established the policy of using air-line respirators which use air from surface for all rock headings. The ventilation system should be such that the exhaust air should not contact workers and this necessitates many raises and crosscuts to be used as by-passes for exhaust air. To improve the ventilation system and other means to protect the health and safety of the employee, our company has spent around $400,000.

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* Chief Engineer, Mining Division, U. S. Bureau of Mines.

† U. S. Bureau of Mines Experiment Station.

‡ General Superintendent, Montreal Mining Co.

Effect of Revaluation on the Gold-mining Industry

Recent Trends in Production, Ore Reserves and Costs

By John J. Croston,* Member A.I.M.E.

(New York Meeting February, 1936)

The year 1935 witnessed one of the world's oldest industries—gold mining—attain the stature of a billion dollar business. Preliminary estimates indicate that somewhat over 30,000,000 oz. of gold was produced, the largest output ever recorded and exceeding the previous year by more than 11 per cent. Of this huge total the Witwatersrand field produced about 10,800,000 oz., compared with 10,480,000 oz. in 1934. Russia was apparently the stellar performer, and from incomplete data it appears that between five and five and a half million ounces were produced by the Soviets compared with 4,200,000 last year and 2,814,000 in 1933. The United States nosed Canada out of third place with a production of 3,546,000 oz. against 2,916,000 in 1934, while Canada increased her total from 2,965,000 to 3,291,000 oz. Similarly, gains of lesser magnitude were made in Australia, the Gold Coast and other countries. As a result of great activity in developing outlying areas on the Rand, and the great number of new companies building mills in Canada, Australia, the Gold Coast and the United States, the years 1936 and 1937 should see new high records, as these new producers will more than offset any decline in the older properties.

The whole world has become "gold conscious" during the past three years. It has been the subject of innumerable articles by economists and others, but, unfortunately, because of their lack of actual mining knowledge, they have treated the production of gold as a static affair, ignoring geological and technical factors.

It is hardly necessary to draw attention to the fact that the world's gold-mining industry was slowly but steadily shrinking in importance, owing to the gradual depletion of known deposits that were profitable to work at costs existing prior to 1929. The International Geological Congress at Pretoria and the League of Nations Committee drew attention to the estimates of the mining experts of the various governments predicting a steady falling off in world production. On the Rand there

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* Engineer, Boston, Mass.

were 10 mines that could not pay dividends in 1929, and the engineer of the Union Government predicted a drop to about 8,000,000 oz. annually by 1936 and to 4,000,000 oz. by 1945, based upon the costs existing at that time and upon the then standard price of 85 shillings. The United States and Australia, both producing at a relatively low rate, were slated for a still greater decline and only Canada gave expectations of increased output. No account was taken at that time of Russian production, as the U.S.S.R. was not giving out information on the annual output or in regard to developments in that republic. It was known, however, that prior to the revolution most of the gold came from placers and little was known about the possibility of large lodes.

The decline in production was based upon the inexorable fact that the deeper you dig and the leaner the ore treated, the higher become the production costs. Several countries offered a bonus on gold production as a means of helping the struggling producers, and this in effect was the same as revaluing the gold as far as its sale by the mines was concerned.

If the world wants gold in sufficient needs for world commerce, it will have to pay a price commensurate with the cost of producing it. Gold increased almost four times in price during the period 1344 to 1717, when it was pegged at the equivalent of $20.67 per troy ounce. It would be ridiculous to attempt to prove that this price bore any relation to the cost of producing gold either in 1717, in 1890 or 1929. It was purely an arbitrary figure, but with the discovery of rich deposits in California, Australia and South Africa was sufficient to bring forth a requisite volume of production. Methods of mining and treatment have reached a high degree of efficiency, and few radical reductions in per-ton costs are to be expected. The paucity of new discoveries, and the necessity for treating lower grades from greater depths indicated a gradual drying up of production unless some stimulus were given to the industry.

This gradual shifting of the economic phases of world gold production has seemingly escaped the attention of the majority of monetary writers. They have treated gold production as a static affair unaffected by changes in reserves, grade, depth and other factors in exploiting this wasting natural asset. Certain phases of the economics of gold mining are not so simple as those of hog raising or wheat growing.

The first fillip to gold mining came with the onset of the depression. Labor became more abundant and efficient, wage rates dropped, as did the cost of supplies, enabling marginal producers to gain a better foothold. The second boost came when Great Britain went off the gold standard, enabling the producers in certain countries to benefit by a substantial exchange premium. The last and greatest step was taken when the United States Government officially revalued gold from $20.67 to $35 per ounce. A careful consideration of the background factors leads to the conclusion that gold will not go back to its gold price.

The new status that gold mining has attained has made it desirable to survey the industry throughout the world to ascertain how revaluation has affected production, ore reserves and costs.

Gold Production

The Witwatersrand field in South Africa, producing well over one-third of the world's gold, has 35 large mines. Two of them jockey for first place as the world's leading gold mine—Crown and Government Areas. Both have produced well over a million ounces of gold each in a single year, a record not even remotely approached by any other mines in the world's history. Eight others produce more than 400,000 oz. annually, while an additional 22 have outputs ranging from 100,000 to 400,000 oz. each. The Rand has 31 of the 53 gold mines of the world producing over 100,000 oz. annually. Naturally developments there are of paramount interest.

During the past year extensions of plant capacity have either been completed or are in progress by a number of producers, and during 1935 a new high-record tonnage was treated. More than 44,500,000 tons were crushed for an average yield of 0.2375 oz. per ton. Several hundred new companies have been floated in the past couple of years but only about 20 are of importance.

The new price of gold has put hundreds of square miles of land into the category of potentially productive ground. Magnetometric surveys and boreholes to the extreme west of the West Rand would indicate that a new Rand is in the making. The 40-mile strip owned by West Witwatersrand Areas, Ltd., a subsidiary of the Consolidated Goldfields of South Africa, Ltd., appears to be underlain by the gold reefs, and it is expected that four or five large companies will be floated to work this area. Western Reefs Exploration & Development Co., Ltd., and other organizations controlled by the big Kaffir finance houses, are actively exploring this new territory. Similarly, to the east of the present Far East Rand exploration has shown the existence of the gold reefs, and this area may also furnish new mines in the near future.

Fourteen new companies are now embarked on large-scale development of their holdings. Six represent an investment of more than $10,000,000 each and the other eight an investment in excess of $5,000,000 each. All of them are sponsored by the great Kaffir mining finance houses, and so highly do the British esteem these organizations that the offerings of a number of them were oversubscribed 20 to 40 times.

The first to commence crushing will be Rand Leases, the only one of the new companies to hold acreage in the Central Rand. It represents a consolidation of the properties of a number of old producers together with a great deal of additional acreage into one large unit between Durban Roodepoort Deep and Consolidated Main Reef. Within the next two

months it will commence milling at an initial rate of 660,000 tons a year. Later in the year Vogelstruisbult will start operating at an initial rate of 600,000 tons a year. Indications point to a good grade of ore and a high percentage of payable ground. This property is in the southeast Rand. The third property to enter production will be Grootvlei, early in 1937, with a milling capacity of 800,000 tons per year. This property adjoins East Geduld and appears to contain a good grade of ore and high percentage of payable ground.

Later South African Land & Exploration and East Daggafontein will start mining and milling, followed by Vlakfontein, Spaarwater, Ventersport, Palmietkuil, Rietfontein No. 11, Marievale Consolidated, Van Dyk, West Spaarwater and Witwatersrand Nigel. East Daggafontein has announced a 1,000,000-ton plant, Venterspost, the sole representative of the West Rand, a treatment plant of 1,000,000 tons per year to be doubled later, and Palmietkuil a plant of 800,000 tons capacity. Work on the others has not progressed to the point where the companies have made official statements as to the size of the proposed milling plants.

The Union Government estimates that these 14 new companies will spend over $110,000,000 on their physical plants, employ 15,000 whites and 135,000 natives; and estimating about the same average grade as the other operating companies this would indicate that these new mines will add about 3,800,000 oz. per year to the Rand output, or more than all the mines in either the United States or Canada produce at the present time.

In the Belgian Congo, Kilo Moto, the great alluvial and lode producer, made a material gain over 1934, when it produced over 211,000 oz., and there was substantial activity by other companies there and in Rhodesia. Tanganyika Territory is enjoying a mild gold boom, and one development, East African Goldfields, Ltd., appears destined to become a substantial producer. On the Gold Coast the old established "jungle" producers had a record year, while a host of new companies have entered the field. Bibiani, having developed a large tonnage, and operated a pilot treatment mill for some time, should soon start larger operations, while Marlu, with nearly 3,000,000 tons developed, is certain to take important rank. Kimingini has started milling in Kenya Colony and others are active in that area.

Turning to our own country, Homestake, soon to celebrate its sixtieth year, maintained its leadership by a wide margin, followed by Alaska-Juneau and Empire-Star, the latter now representing a number of properties in the Grass Valley region. U.S. Smelting in Alaska and Natomas and Yuba in California remain the leading dredging enterprises. Idaho-Maryland has forged its way along, while numerous other small enterprises in California, Nevada, Montana and Colorado have in the aggregate produced a large total. Of great interest during the year was the start of

operations at the Golden Queen mine at Mojave, in California, representing perhaps the only really important gold discovery in this country in about 30 years. Only three lode mines and one dredging enterprise in the United States, including Alaska, can be included in the 53 mines of the world producing over 100,000 oz. per year. The rest of the production comes from a great number of small enterprises and as a byproduct of the production of other metals.

An all-time high was reached in Canadian production in 1935. While the "Big Six" continue to produce over half of the Dominion's gold, many new mines have entered operations, and doubtless this proportion will change in the next few years. During 1934 Central Patricia, McWatters, Northern Empire and Little Long Lac started milling, to mention but a few of the more important. During the past year Canadian Malartic, McKenzie Red Lake, God's Lake, Lamaque and Pickle Crow were the most important, and others are scheduled for the current year. While most of these mines are small operations at present, measured by world standards, a few of them will doubtless attain important size in time, and in the aggregate their production adds very materially to the Dominion's total.

In Brazil, St. John del Rey, opened in 1830, continued large-scale operations, and its Espirito Santo has developed a large tonnage, which should soon swell the total from the old Morro Velho property. In Colombia, Asnazu started dredging operations, and San Nicolas is scheduled for production early this year. The older properties, Frontino, South American Gold & Platinum and Pato, were busy. New Goldfields and Botanamo, in Venezuela, operated steadily and plans were afoot to start operations at Amarilla and El Callao.

No information is yet available regarding 1935 operations at Boliden, in northern Sweden. In France gold mining continued on a small scale, and several gold mines were modernized in Transylvania. Russian production increased tremendously, but no data are available regarding operations at individual properties. The Kolar gold field, in India, added its usual quota, while in Korea Oriental Consolidated and Chosen were in steady production. In Japan most of the gold comes from copper-mining operations, but the Konomai and Taio gold mines were active.

Western Australia experienced a great gold boom in the past two years and hundreds of new companies have been formed. London and South African finance houses have entered "down under" on a substantial scale. The old Mount Morgan is now in steady operation again, and the old Lancefield and Edjudina properties are now milling. Others to enter the producing list are Mount Magnet, New Occidental, Triton, Buninyong and Talbot Alluvials. The older producers maintained heavy outputs although hampered by a strike earlier in the year. Mount Coolon was shut down part of the year. At Yellowdine great activity has

TABLE 1.—*Annual Output of World's Leading Gold Mines,* 1929–1935

IN FINE OUNCES, TROY

Rank	Company	Location	Calendar Years, Unless Otherwise Noted						
			1935	1934	1933	1932	1931	1930	1929
1	Crown	Transvaal, South Africa	981,102	1,001,618	1,043,500	1,042,064	986,329	924,298	856,005
2	Government Areas	Transvaal, South Africa	857,919	915,851	1,033,687	1,146,140	1,129,872	1,007,095	1,086,163
3	Randfontein	Transvaal, South Africa	744,743	730,936	828,782	821,334	745,313	652,606	595,249
4	Homestake	South Dakota, U.S.A.	548,289[a]	471,749	510,968	479,138	431,860	406,000	315,300
5	New Modder	Transvaal, South Africa	536,576	554,118	617,691	770,535	805,560	862,506	873,294
6	East Rand Prop.	Transvaal, South Africa	507,196	477,712	483,130	502,347	501,085	491,095	461,603
7	Sub Nigel	Transvaal, South Africa	475,261	424,281	427,581	390,723	352,624	304,915	252,608
8	Lake Shore	Ontario, Canada	463,225[b]	472,768	499,800	610,463	533,757	377,831	293,587
9	New State	Transvaal, South Africa	444,547	440,678	494,532	511,369	479,205	445,938	391,430
10	Springs	Transvaal, South Africa	435,159	421,799	438,954	455,436	413,688	408,250	398,965
11	Hollinger	Ontario, Canada	418,266	434,257	481,279	499,648	487,123	494,532	455,094
12	Daggafontein	Transvaal, South Africa	409,680	323,925	244,848	177,542	y	y	y
13	Brakpan	Transvaal, South Africa	402,585	404,411	435,054	449,520	414,539	391,852	399,246
14	East Geduld	Transvaal, South Africa	360,931	331,492	292,209	224,603	56,003	y	y
15	West Rand Cons.	Transvaal, South Africa	358,201	339,588	293,496	299,192	287,225	290,671	266,116
16	Geduld	Transvaal, South Africa	324,698	324,135	323,126	325,803	323,616	322,884	318,994
17	Consol. Main Reef	Transvaal, South Africa	320,340	279,366	285,107	285,742	271,140	258,856	263,591
18	Robinson Deep	Transvaal, South Africa	302,690	301,017	319,274	337,696	334,457	365,781	253,147
19	Boliden	Northern Sweden		264,187	238,948	128,605	56,900[c]	27,766[c]	4,064
20	Noranda	Quebec, Canada	268,333	248,615	284,675	341,350	253,363	117,393	68,732
21	Simmer and Jack	Transvaal, South Africa	258,399	245,164	248,208	260,083	262,548	250,485	224,796
22	Modder East	Transvaal, South Africa	244,119	234,495	242,198	261,646	255,908	242,149	243,422
23	McIntyre Porcupine	Ontario, Canada	243,486[b]	239,099	261,529	261,725	229,413	226,266	206,628
24	City Deep	Transvaal, South Africa	238,041	236,210	256,122	257,110	264,018	306,425	295,615
25	Kilo Moto	Belgian Congo, Africa	234,383	211,764	195,392	194,933	170,016	147,799	128,766
26	Van Ryn Deep	Transvaal, South Africa	230,774	220,416	246,400	261,855	268,940	297,602	294,909
27	Wright Hargreaves	Ontario, Canada	215,102[b]	218,203	177,190	171,299	140,520	117,455	83,631
28	Dome	Ontario, Canada	206,795	206,163	218,485	195,111	169,686	37,416[d]	173,042
29	Ashanti[e]	Gold Coast, Africa	199,650	190,797	177,143	175,063	169,360	144,548	118,136
30	Nourse	Transvaal, South Africa	198,671	200,607	218,202	251,869	248,329	223,198	210,757
31	Langlaagte	Transvaal, South Africa	188,804	197,536	249,758	306,390	317,659	326,412	313,105
32	Modder "B"	Transvaal, South Africa	183,024	192,435	210,268	255,685	264,895	293,517	302,724
33	West Springs	Transvaal, South Africa	180,443	173,514	205,876	225,481	210,498	218,054	221,895
34	Balatoc	Philippine Islands		134,161	113,921	101,145	68,085	61,001	56,965
35	Witwatersrand Gold	Transvaal, South Africa	154,905	151,793	145,642	157,617	146,423	145,355	134,984
36	Modder Deep	Transvaal, South Africa	151,979	192,537	220,203	258,354	268,790	275,251	280,665
37	Durban Roodepoort Deep	Transvaal, South Africa	149,978	151,774	169,287	188,671	183,020	172,562	164,809

y Mine not in operation or not yet discovered.
[a] Estimated from annual report.
[b] Local government statistics.
[c] Estimated from government figures.
[d] Mill burned down Oct. 29, 1929.
[e] Year ended Sept. 30.

TABLE 1.—(*Continued*)

Rank	Company	Location	Calendar Years, Unless Otherwise Noted 1935	1934	1933	1932	1931	1930	1929
38	Geldenhuis Deep...	Transvaal, South Africa	148,915	162,301	181,884	203,841	198,843	186,236	176,170
39	Teck-Hughes	Ontario, Canada	140,053[b]	167,412	220,036	282,882	294,422	260,775	243,745
40	Lake View & Star[f]..	Western Australia	133,811	152,835	170,177	93,476	109,261	132,578	118,568[g]
41	Rose Deep.........	Transvaal, South Africa	130,495	137,585	144,025	154,641	153,028	155,841	143,556
42	Bulolo (dredging)...	New Guinea	124,359	106,061	90,379	52,962	y	y	y
43	New Kleinfontein ..	Transvaal, South Africa	122,372	113,707	116,166	123,588	124,782	135,192	135,358
44	Benguet...........	Philippine Islands		113,428	107,157	100,852	79,605	84,998	73,246
45	Alaska-Juneau.....	Alaska	118,998	128,015	150,967	151,578	179,532	163,312	164,993
46	Empire Star[h].......	California, U.S.A.		117,618	107,611	97,005	97,917	79,800	x
47	Witwatersrand Deep	Transvaal, South Africa	118,810	116,442	129,921	128,826	103,779	113,689	113,714
48	Luipaard's Vlei.....	Transvaal, South Africa	115,307	107,500	98,583	101,629	94,471	85,634	70,984
49	Cam and Motor....	Southern Rhodesia, Africa	111,459	107,451	111,878	119,235	123,339	128,656	140,330
50	Nundydroog.......	Mysore, India	111,157	110,536	114,290	89,605	79,836	78,746	80,120
51	Wiluna[i]...	Western Australia	108,501	126,493	124,531	123,085	58,988	y	y
52*	Van Ryn Gold.....	Transvaal, South Africa	106,975	95,910	120,618	129,349	122,488	119,024	112,870
53	Mysore	Mysore, India	94,440	92,507	91,524	88,598	96,042	101,904	102,228
54	St. John del Rey ...	Minas Geraes, Brazil	93,839	96,857	105,899	109,202	115,473	123,161	108,879
55	Pioneer[i]...........	British Columbia, Canada	87,030	83,827	53,548	32,567	16,931	7,850	8,676
56	Nigel.............	Transvaal, South Africa	73,914	20,963	y	y	y	y	y
57	Champion Reef.....	Mysore, India	68,068	66,400	64,056	66,038	65,719	62,660	66,562
58	Yuba Cons.[j] (dredging).	California, U.S.A.	67,096	50,457	56,160	52,542	48,730	59,981	106,673
59	Natomas (dredging)	California, U.S.A.	65,296	59,437	69,425	62,544	58,793	66,251	59,660
60	Siscoe.............	Quebec, Canada	64,988	63,394	54,810	48,683	35,936	17,768	14,874
61	Grand Lacs Africaines (Alluvial).	Belgian Congo, Africa	59,383	51,024	49,577	43,018	36,910	31,315	32,441
62	South American Development.	Zaruma, Ecuador	k						
63	Martha (formerly Waihi)...........	New Zealand	58,438	63,336	69,185	73,593	75,468	78,680	73,187
64	Globe and Phoenix	Southern Rhodesia, Africa	56,221	59,938	65,204	75,092	65,865	65,961	60,310
65	New Goldfields of Venezuela.	Bolivar, Venezuela	55,929	54,004	28,626	24,425	25,231	12,488	11,700
66	Great Boulder......	Western Australia	54,997	65,842	76,239	79,186	79,547	80,207	87,554
67	Sylvanite..........	Ontario, Canada	54,739[b]	50,337	44,607	39,919	43,437	38,303	33,168
68	Transvaal Gold.....	Transvaal, South Africa	51,880	57,127	63,803	64,789	60,295	58,544	57,988
69	Ooregum..........	Mysore, India	52,344	50,303	51,316	52,223	63,083	63,700	82,483
70	Frontino[f]..........	Antioquia, Colombia	49,970	52,232	52,275	46,649	37,130	18,492	15,939
71	Taquah and Abosso.	Gold Coast, Africa	49,408	49,952	44,453	39,888	42,922	44,946	41,046
72	Salsigne[f]..........	Dept. Aude, France	47,436	48,084	48,098	34,488	28,808	24,338	22,763
73	Idaho-Maryland....	California, U.S.A.	73,313	43,656	41,908	47,379	27,797	11,662	x
74	Bralorne..	British Columbia, Canada	47,089	45,296	25,885	22,234	y	y	y
75	Beattie............	Quebec, Canada		52,400	22,598	y	y	y	y

x No data received.

* The 53 mines of over 100,000 oz. annual output include the dredging enterprises of the U. S. Smelting, Refining & Mining Co. in Alaska. Their production is not given separately but included in total of that company.

[f] Year ended June 30.

[g] 16 months ended June 30.

[h] Unofficial estimates.

[i] Year ended March 30.

[j] Year ended Feb. 28.

[k] No data released. Approximate importance shown by relative rank.

TABLE 1.—(*Continued*)

Rank	Company	Location	Calendar Years, Unless Otherwise Noted						
			1935	1934	1933	1932	1931	1930	1929
76	South American Gold and Platinum.	Andagoya, Colombia	44,846[m]	28,134	25,965	18,285	9,736	9,048	8,054
77	Taio Gold..........	Fukuoka, Japan		49,500	*x*	59,512	34,780	35,211	35,583
78	Simau..............	Netherlands East Indies		43,000	46,600	46,153	54,168	54,702	48,455
79	Chosen[i]............	Chosen (Korea)	42,453	40,116	39,547	38,010	39,349	32,900	29,453
80	Antamok...........	Philippine Islands		17,999	6,159	*y*	*y*	*y*	*y*
81	United Gold Mines..	Colorado, U.S.A.	39,165	31,029	22,575	23,839	28,794	37,946	34,124
82	Oriental Cons.[j].....	Chosen (Korea)		41,251	53,180	46,162	54,987	50,684	48,500
83	Wanderer..........	Southern Rhodesia, Africa	38,872	37,109	41,725	41,471	42,248	44,580	4,381
84	Sumitomo Konomai.	Japan			38,000	42,865	28,935	27,464	24,384
85	Howey.............	Ontario, Canada	37,753	45,985	40,460	53,948	41,702	22,147	*y*
86	New Guinea Goldfields[e]..	New Guinea	37,694	51,684	45,005	43,205	*y*	*y*	*y*
87	Mountain..........	California, U.S.A.	37,199	16,540	19,902	24,088	14,672	*x*	*x*
88	London............	Colorado, U.S.A.							
89	Itogon.............	Philippine Islands		31,764	33,500	*y*	*y*	*y*	*y*
90	Sons of Gwalia......	Western Australia	35,767	42,740	43,033	43,568	41,441	38,225	30,929
91	Ariston[e]............	Gold Coast, Africa	34,261	38,500	42,100	25,997	25,123	*y*	*y*
92	Mount Morgan[f]....	Queensland, Australia	35,285	30,392	14,550	26,245	*y*	*y*	*y*
93	Cresson Cons.......	Colorado, U.S.A.	34,013	35,827	39,585	37,586	31,017[n]	32,276[n]	44,053[n]
94	Eastern Transvaal Cons.	Transvaal, South Africa		35,200	28,400	26,365	26,688	34,780	30,200
95	Capital (dredging)..	California, U.S.A.		32,200	32,500	31,500	27,000	17,150	12,520
96	Coniaurum.........	Ontario, Canada	32,417	28,436	33,596	41,582	36,278	35,664	30,641
97	San Antonio........	Manitoba, Canada	32,152	21,638	22,778	17,401	*y*	*y*	*y*
98	Premier............	British Columbia, Canada	31,747	38,371	51,863	78,716	82,394	90,084	98,442
99	Glyns Lydenburg...	Transvaal, South Africa	31,633	35,085	33,767	31,021	29,218	25,600	24,594
100	Little Long Lac.....	Ontario, Canada	31,429	2,457	*y*	*y*	*y*	*y*	*y*
101	Golden Plateau[f]....	Queensland, Australia	31,375	17,174	*y*	*y*	*y*	*y*	*y*
102	Kushikino..........	Kagashima, Japan				30,935	31,121	29,818	29,642
103	Macassa...........	Ontario, Canada	30,589	32,056	3,725	*y*	*y*	*y*	*y*
104	Amalgamated Banket Areas.	Gold Coast, Africa		24,591	*y*	*y*	*y*	*y*	*y*
105	Buffalo-Ankerite....	Ontario, Canada	29,124	20,503	22,343	10,602	*y*	42	3,458
106	Lamaque...........	Quebec, Canada	29,123	*y*	*y*	*y*	*y*	*y*	*y*
107	Boulder Perseverance	Western Australia		33,053	45,000	39,541	45,500	42,101	52,818[p]
108	Bibiani[e]............	Gold Coast, Africa	26,063	18,793	3,201	*y*	*y*	*y*	*y*
109	Pickle Crow........	Ontario, Canada	25,107	*y*	*y*	*y*	*y*	*y*	*y*
110	South Kalgurli......	Western Australia	24,690	30,995	43,155	49,631	44,900	34,281	45,180
111	Yukon Cons. (Dredging)...............	Yukon, Canada	23,429	27,500	30,000	35,000	37,000	29,500	27,000
112	Kirkland Lake Gold.	Ontario, Canada	22,215	20,316	18,465	25,323	28,315	25,764	16,999
113	Central Patricia....	Ontario, Canada	22,061	6,373	*y*	*y*	*y*	*y*	*y*
114	North Kalgurli.....	Western Australia		23,215	*x*	23,487	8,099	5,113	3,464
115	Redjang Lebong....	Netherlands East Indies		23,000	26,882	26,801	27,042	28,451	28,528
116	Reno..............	British Columbia, Canada	21,586	15,979	11,551	*q*	9,751	7,876	1,790
117	Original Sixteen-to-One	California, U.S.A.	*k*						

[m] Crude gold.
[n] Year ended Aug. 31.
[p] 15 months ended Dec. 31.
[q] Shutdown. Mill destroyed by fire.

TABLE 1.—(*Continued*)

Rank	Company	Location	Calendar Years, Unless Otherwise Noted						
			1935	1934	1933	1932	1931	1930	1929
118	Raub Australian[j]	Federated Malay States	20,858	26,678	26,259	26,136	24,727	23,362	18,340
119	Toburn	Ontario, Canada	20,200	20,401	23,020	9,841	y	y	y
120	Young-Davidson	Ontario, Canada	20,382	3,878	y	y	y	y	y
121	Botanamo	Bolivar, Venezuela		19,851	22,913	24,870	25,934	26,595	25,650
122	Blackwater	New Zealand	19,723	14,966	20,813	22,649	19,265	16,502	15,160
123	Equatoriale	French Congo, Africa	19,722	20,055	20,723	15,456	8,855	2,254	y
124	Argonaut	California, U.S.A.	19,558	21,357	27,086	23,762	x	x	x
125	Rezende	Southern Rhodesia, Africa	18,812	18,307	26,691	31,348	32,557	32,300	34,957
126	Belgikaor	Belgian Congo, Africa		16,800	10,225	4,475	1,267	241	y
127	Lava Cap	California, U.S.A.	24,459	6,514	y	y	y	y	y
128	Lancefield	Western Australia	18,596		y	y	y	y	y
129	Northern Empire	Ontario, Canada	18,437	5,663	y	y	y	y	y
130	Sherwood Starr	Southern Rhodesia, Africa	17,758	20,261	22,186	22,964	27,450	35,746	24,970
131	Cariboo	British Columbia, Canada	17,458	10,967	7,395	y	y	y	y
132	Island Mountain	British Columbia, Canada	17,435	1,591	y	y	y	y	y
133	McWatters	Quebec, Canada	17,078	2,961	y	y	y	y	y
134	Carson Hill[e]	California, U.S.A.	16,930	x	y	y	y	y	y
135	Paymaster	Ontario, Canada	16,799	2,021	y	y	y	y	y
136	Cuthbert's Misima[s]	Misima Island, Oceania	16,712	16,546	y	y	y	y	y
137	Tele	Belgian Congo, Africa			15,883	12,860	7,041	6,977	5,369
138	Lonely Reef	Southern Rhodesia, Africa	16,626	19,024	23,569	27,603	38,297	45,937	49,368
139	McKenzie Red Lake	Ontario, Canada	15,235	y	y	y	y	y	y
140	Baguio	Philippine Islands		13,126	3,131	y	y	y	y
141	Rimu (dredging)	New Zealand		14,965[m]	14,673	14,512	8,131	9,840	11,500
142	New Pioneer Central Rand[f].	Transvaal, South Africa	15,189	y	y	y	y	y	y
143	Cardinal	California, U.S.A.	14,586	y	y	y	y	y	y
144	Panique	Philippine Islands		10,923	y	y	y	y	
145	Kennedy	California, U.S.A.	[k]						
146	Ipo	Philippine Islands		11,866	y	y	y	y	y
147	Eureka-Standard	Utah, U.S.A.	12,483	26,240	27,043	41,742	48,207	30,541	14,755
148	Pato Cons.[s] (dredging)	Colombia	14,527[r]	x	30,460	24,984	25,439	27,483	30,565
149	Central Eureka	California, U.S.A.							
150	Maroc[t]	Sierra Leone, Africa	13,661	13,605	10,337	8,034	y	y	y
151	Sullivan Cons.	Quebec, Canada	13,432	4,062	y	y	y	y	y
152	Rhodesian Corp.[t] "Fred"	Southern Rhodesia, Africa	12,762	12,881	12,784	13,094	14,012	19,778	21,181
153	Norseman[u]	West. Australia	12,451	10,666	y	y	y	y	y
154	Ashley	Ontario, Canada	12,358	13,181	17,313	5,114	y	y	y
155	Dentonia	Brit. Col., Can.	12,027	4,380	y	y	y	y	y
156	Anglo-Huronian	Ontario, Canada	11,970	15,541	24,245	21,939	27,236	43,883	39,569
157	Central Manitoba[s]	Manitoba, Can.	11,302	12,108	19,447	19,951	19,781	20,379	24,654
158	Matachewan Cons.	Ontario, Canada	10,113	679	y	y	y	y	y
159	Mount Coolon	Queensld., Austr.		32,820	38,444	5,951	y	y	y
160	Veraguas	Panama	5,064	14,979	y	y	y	y	y

[r] Eleven months to April 30.
[s] Year ended April 30.
[t] Year ended July 31.
[u] Year ended Oct. 31.

TABLE 2.—*Gold Production of Custom Smelting, Silver and Base-metal Mining Companies*

(1935 Figures Included Where Reports Are Available)

Company	1935	1934	1933	1932	1931	1930	1929
Amer. Smelt. & Ref. Co	1,869,299	1,476,487	1,298,697	1,345,960	1,454,774	1,760,702	1,461,215
Amer. Metal Co. Ltd	578,960	414,284	335,371	321,535	522,902	485,176	478,504
U.S. Smelt., Ref. & Min. Co	356,966	321,385	328,826	350,678	314,755	322,968	266,197
Golden Cycle Corp	158,941	162,598	145,271	158,522	122,711		
Hudson Bay M. & S. Co. Ltd	104,218	99,334	94,745	82,565	73,000		
Internat. Nickel Co.[d]	69,944	74,375	21,335	22,675	23,381	22,867	7,802
Hidachi mine (Nippon Mining Co.)				81,953	84,063	79,753	64,913
Saganoseki mine (Nippon Mining Co.)				68,514	122,899	98,333	76,222
Phelps Dodge Corp	107,435	68,889	43,882	28,736	99,749	70,033[a]	100,607[a]
Cia. Minera las dos Estrellas					80,153	78,417	62,920
Utah Copper Co	67,645	44,069	39,264	25,399	54,124	64,240	116,087
San Luis Mining Co					46,555	43,919	40,800
Naoshima mine (Mitsubishi Mining Co.)				39,262	36,166	30,805	23,632
Cons. Min. & Smelt. Co. Can	65,131	35,328	22,393	33,346	24,968	25,782	14,694
Besshi (Sumitomo Mining Co.)				23,567	38,696	28,662	24,467
Cerro de Pasco Copper Corp	28,000	20,000	20,326	12,000	28,670	29,778	33,671
Cia. Dos Carlos, S.A.					24,403	31,090	29,354
Cia. Min. y Beneficiadore de Inde					22,442	25,560	16,691
Greene Cananea Copper Co	16,370	20,851	13,528	9,596	8,447	7,941	12,366
Kosaka mine (Fujita Mining Co.)				17,355	18,558	18,349	17,500
Anaconda Copper Mining Co		49,208	40,886	18,551[c]	32,724	21,209[c]	61,980
Mexican Corp.[b]	20,460	13,193	6,744	8,339	9,195	7,941	9,838
Howe Sound Co.[d]	10,690	12,034	13,701	8,868	5,476	12,770	14,252
New York & Honduras Rosario	12,274	12,996	17,210	16,054	15,835	13,498	10,304
Guanajuato Red. & Mines Co		11,810	12,482	15,065	16,526	11,382	12,346
Magma Copper Co	32,899	30,311	11,255[c]	8,126[c]	9,244[c]	10,154[c]	13,405[c]
Union Min. du Haut Katanga		20,255	9,391				
Bunker Hill & Sullivan		5,230[c]	11,101[c]	34,760	44,811	46,000	
Andes Copper Mining Co	13,351	10,459	6,626	1,991	4,791	6,629	16,537
United Verde Copper Co		shut	shut	shut	10,350	42,937	62,097
Nevada Cons. Copper Co				8,311	26,670	14,537	14,536

[a] Calumet & Arizona output included, although not merged during these years.
[b] Year ended June 30. Cyanide plant operations only.
[c] Own ores only, custom ores not included.
[d] Annual sales.

been manifest, because of the remarkably rich ore uncovered during development, and milling is scheduled for the present summer (winter down there). Rich ore has been developed at Loloma and the Emperor in the Fiji Islands. Bulolo plans two new dredges for its New Guinea placer, one to dig 115 ft. below the water line.

In the Philippines, Balatoc and Benguet were the principal producers, with Antamok, Baguio, Suyoc, Panique, United Paracale and other newer properties adding to the output.

Table 1 gives the annual output in fine ounces of the leading gold mines of the world from the start of the depression in 1929 to the end of 1935. It is believed to be the most complete table ever made of the world's gold producers. The figures in practically all instances have been obtained from company reports, some from local governmental statistics,

while a very few have been estimated from the production data. It will be noted that while the price of gold rose substantially in 1933–34 production fell away. This seeming paradox of an increased offering price serving to reduce rather than increase production was unforeseen by the brokerage statisticians that wore out dozens of pencils tabulating the prospective earnings of the gold shares with gold at $25, $30, $35, $40 and more per ounce. The answer lies in the policy of treating lower grade ore, thus increasing the tonnage available for milling and extending the life of the property. This policy in most cases made it possible to pay out a smaller percentage of earnings in excess profits or other taxes, and in some cases has been practically forced upon the mines by taxation based on the grade treated, resting heaviest on the rich producing mines. This lag in production was but a temporary phenomenon, as many properties decided to treat larger tonnages, and the effect of this increase in milling capacity was shown in 1935 and will continue throughout the current year.

"Reported" Ore Reserves

Practically all of the gold mines having shares listed on the various stock exchanges make periodical reports of the tonnage and value of the gold ore blocked out. It was considered desirable to get a rough quantitative estimate of the effect of revaluation upon the reported ore reserves of the world's more important mines and alluvial properties. No attempt was made to include the gold reserves of the great copper mines now developed (with the sole exception of Noranda), which probably amount to about 20,000,000 oz., nor were the great Mexican silvers or other silver, lead, zinc, nickel or other deposits that carry gold values included. The ore reserves of these mines would not be appreciably affected by revaluation, so that there would be little change to be expected in the 1932 and 1935 totals. These "reported" ore reserves are the tonnages fully developed underground and should not be confused with the actual reserves, which are in most cases many times greater but not usually determinable without complete development of the deposit. In connection with some of the great gold-bearing copper deposits, dredging enterprises, and certain gold deposits (as on the Rand) the total amount remaining to be mined to the exhaustion of the deposit is reasonably well known in advance. In underground operations it is rarely feasible technically or advisable economically to block out in advance tonnages that would not be mined for many years to come. Many of the world's greatest gold mines active over a long period of years have never had over a few years of developed and reported reserves. Cam & Motor is an instance of this—five years ago it reported about 875,000 tons; since then it has treated about 1,500,000 tons and still is able to report over 1,230,000 tons of reserves.

TABLE 3.—"*Reported" Gold Ore Reserves of World's Principal Mines Grouped by Principal Producing Regions*

For Latest Fiscal Period and for Period before Revaluation of Gold

Company	Year Ended	Reported Ore Reserves, Tons	Average Gold Content, Oz. per Ton	Total Metallic Content of Gold, Troy Oz.	Year Ended	Reported Ore Reserves, Tons	Average Gold Content, Oz. per Ton	Total Metallic Content of Gold, Troy Oz.
South Africa								
Witwatersrand Field Total Ore Reserves Reported by the Large Established Producers	June 30, 1935	171,750,690	0.28	48,090,193	Dec. 31, 1932	90,860,000	0.3525	32,028,150
Crown Mines, Ltd.	Dec. 31, 1934	21,342,390	0.2925	6,242,649	Dec. 31, 1932	13,742,950	0.332	4,562,659
Government Gold Mining Areas (Modderfontein) Consolidated, Ltd.	Dec. 31, 1934	11,782,000[a]	0.365	4,300,430	Dec. 31, 1932	9,860,000[a]	0.445	4,387,700
Springs Mines, Ltd.	Dec. 31, 1934	10,007,871	0.314	3,142,472	Dec. 31, 1932	3,838,520	0.442	1,696,626
Randfontein Estates Gold Mining Co. (Witwatersrand) Ltd.	Dec. 31, 1934	13,007,000[a]	0.235	3,056,645	Dec. 31, 1932	7,094,000[a]	0.315	2,234,610
New Modderfontein Gold Mining Co., Ltd.	June 30, 1935	9,305,300	0.265	2,465,904	June 30, 1932	7,294,700	0.375	2,735,513
Geduld Proprietary Mines, Ltd.	Dec. 31, 1934	9,000,000	0.270	2,430,000	Dec. 31, 1932	5,800,000	0.330	1,914,000
East Geduld Mines, Ltd.	Dec. 31, 1934	7,000,000	0.330	2,310,000	Dec. 31, 1932	3,900,000	0.355	1,384,500
East Rand Proprietary Mines, Ltd.	Dec. 31, 1934	8,458,300	0.255	2,156,867	Dec. 31, 1932	4,059,750	0.313	1,270,702
West Rand Consolidated Mines, Ltd.	Dec. 31, 1934	9,271,000	0.225	2,085,975	Dec. 31, 1932	5,018,000	0.265	1,329,770
Consolidated Main Reef Mines & Estate, Ltd.	June 30, 1935	7,948,300	0.239	1,899,644	June 30, 1932	3,690,720	0.3194	1,178,816
Daggafontein Mines, Ltd.	Dec. 31, 1934	4,873,211	0.334	1,627,653	Dec. 31, 1932	2,101,980	0.385	809,262
New State Areas, Ltd.	Dec. 31, 1934	4,212,000[a]	0.365	1,537,380	Dec. 31, 1932	2,746,000[a]	0.445	1,221,970
Brakpan Mines, Ltd.	Dec. 31, 1934	4,879,870	0.3165	1,544,479	Dec. 31, 1932	2,896,250	0.388	1,123,745
Robinson Deep, Ltd.	Dec. 31, 1934	5,916,000	0.245	1,449,420	Dec. 31, 1932	3,076,000	0.290	892,040
Sub Nigel, Ltd.	June 30, 1935	1,818,000	0.780	1,418,040	Dec. 31, 1932	1,480,000	0.880	1,302,400
Modder East, Ltd.	June 30, 1935	5,242,900	0.236	1,237,324	June 30, 1932	2,245,100	0.290	651,079
Durban-Roodepoort Deep, Ltd.	Dec. 31, 1934	4,258,800	0.2716	1,156,690	Dec. 31, 1932	2,753,900	0.320	881,245
West Springs, Ltd.	Dec. 31, 1934	4,176,790	0.246	1,027,490	Dec. 31, 1932	2,765,070	0.296	818,461
Nourse Mines, Ltd.	June 30, 1935	4,291,000	0.234	1,004,094	June 30, 1932	1,861,800	0.295	549,231
City Deep, Ltd.	Dec. 31, 1934	3,496,400	0.270	944,028	Dec. 31, 1932	2,167,200	0.3155	683,519
Van Ryn Deep, Ltd.	Dec. 31, 1934	3,384,000[a]	0.210	710,640	Dec. 31, 1932	2,500,000[a]	0.300	750,000
Modder "B" Gold Mines, Ltd.	Dec. 31, 1934	3,510,910	0.1981	695,511	Dec. 31, 1932	1,125,870	0.3292	370,634
Simmer & Jack Mines, Ltd.	Dec. 31, 1934	2,735,700	0.250	683,925	Dec. 31, 1932	1,645,300	0.300	493,590
Primrose Gold Mining Co. (1934) Ltd.	Dec. 31, 1934	2,584,000	0.250	646,000				
Luipaard's Vlei Estate & Gold Mining Co., Ltd.	June 30, 1935	2,593,130	0.2433	630,909	June 30, 1932	1,095,400[a]	0.265	290,281
Rand Leases (Vogelstruisfontein) Gold Mining Co., Ltd.	Dec. 31, 1935	2,928,000	0.2338	684,566				
Rose Deep, Ltd.	Dec. 31, 1934	2,058,900	0.190	391,191	Dec. 31, 1932	1,024,100	0.2331	238,685
New Kleinfontein Co., Ltd.	Dec. 31, 1934	1,270,858	0.2545	323,433	Dec. 31, 1932	431,000	0.245	105,595
Geldenhuis Deep, Ltd.	Dec. 31, 1934	1,578,300	0.204	321,973	Dec. 31, 1932	618,800	0.2726	168,725
Witwatersrand Deep, Ltd.	Dec. 31, 1934	1,085,900	0.265	287,764	Dec. 31, 1932	596,000	0.310	184,760
Langlaagte Estate & Gold Mining Co., Ltd.	Dec. 31, 1934	1,385,000[a]	0.195	270,075	Dec. 31, 1932	850,000[a]	0.345	293,250
Modder Deep Levels, Ltd.	Dec. 31, 1934	900,000	0.230	207,000	Dec. 31, 1932	1,200,000	0.413	495,600
Witwatersrand Gold Mining Co., Ltd. ("Knights")	Dec. 31, 1934	1,204,000[a]	0.165	198,660	Dec. 31, 1932	349,000[a]	0.235	82,015
Nigel Gold Mining Co., Ltd.	Dec. 31, 1934	519,368	0.352	182,818				
Glyns-Lydenburg, Ltd.	July 31, 1935	452,400	0.380	171,912	July 31, 1932	330,970	0.395	130,733

Van Ryn Gold Mines & Estate, Ltd	June 30, 1935	1,187,680	0.140	166,275	June 30, 1932	490,491	0.205	100,551
Transvaal Gold Mining Estates, Ltd	Mar. 31, 1935	516,600	0.295	152,397	Mar. 31, 1932	386,000	0.395	152,470
Eastern Transvaal Cons. Mines, Ltd	June 30, 1935	676,790	0.1799	121,777				
Village Main Reef, Ltd	June 30, 1935	500,000	0.22	110,000				
Witburg Exploration & Development Co., Ltd. (Kosterfontein)	June 30, 1935	400,000	0.250	100,000				
Jumpers Consolidated Gold Mining Co., Ltd	Dec. 31, 1934	400,000	0.200	80,000				
Rietfontein Consolidated Mines, Ltd	Mar. 31, 1935	256,176	0.315	80,695				
Klerksdorp Consolidated Goldfields, Ltd	Oct. 1, 1935	197,000	0.300	59,100				
Carrig Gold Mines, Ltd	June 30, 1935	332,000	0.117	38,800				
New Pioneer Central Rand Gold Mining Co., Ltd	June 30, 1935	176,000	0.2165	38,104				
Golden Ridge Mining Co. (Pty) Ltd	June 30, 1935	200,000	0.162	32,400				
Eastern Star Gold Mines, Ltd	Dec. 31, 1934	60,000	0.37	22,200				
Belgeric Gold Mining Co., Ltd	Dec. 31, 1934	68,426	0.280	19,159				
Great Western Gold Mining Co., Ltd	Sept. 30, 1935	33,860	0.2695	9,125				
Aurora West Consol. Gold Mng. Co., Ltd	Oct. 31, 1935	2,999,000						
New Wolhuter Gold Mines, Ltd	Dec. 31, 1935	1,250,000						
Rhodesia								
Cam & Motor Gold Mining Co. (1919) Ltd	June 30, 1935	1,233,000	0.460	567,180	June 30, 1932	1,000,000	0.5176	517,600
Wanderer Consolidated Gold Mines, Ltd	June 30, 1935	776,500	0.185	143,653	June 30, 1932	431,000	0.250	107,750
Globe & Phoenix Gold Mining Co., Ltd	June 30, 1935	124,800	1.125	140,400	Dec. 31, 1932	108,100	1.235	133,500
Sherwood-Starr Gold Mining Co., Ltd	June 30, 1935	383,000	0.310	118,730	June 30, 1932	246,000	0.5235	128,781
Phoenix Prince Gold Mining Co., Ltd	June 30, 1935	226,267	0.360	81,456				
Bushtick Mines (1934) Ltd	June 30, 1935	344,050	0.2225	76,551				
Rhodesian Corp., Ltd. Fred Mine	Sept. 30, 1935	107,417	0.435	46,726				
Rezende Mines, Ltd	Dec. 31, 1934	75,000	0.345	25,875	Dec. 31, 1932	119,000	0.410	48,790
Bernheim Gold Mining Co., Ltd	Dec. 31, 1934	45,300	0.375	16,987				
Other African								
Ashanti Goldfields Corp., Ltd	Sept. 30, 1935	1,059,224	0.9923	1,051,058	Sept. 30, 1932	642,100	1.180	757,678
Bibiani (1927) Ltd	Sept. 30, 1935	1,706,200	0.415	708,073	Sept. 30, 1932	290,000	0.525	152,250
Marlu Gold Mining Areas, Ltd	Mar. 31, 1935	2,812,000	0.230	646,760				
Ariston Gold Mines (1929) Ltd	Sept. 30, 1935	1,032,900	0.4803	496,124	Sept. 30, 1932	530,000	0.5227	277,031
Taquah & Abosso Mines, Ltd	Mar. 31, 1935	913,590	0.351	320,670	Mar. 31, 1932	355,965	0.366	130,283
Amalgamated Banket Areas, Ltd	Aug. 1, 1935	830,237	0.331	274,808				
Konongo Gold Mines, Ltd	Sept. 30, 1935	140,327	0.465	65,252				
Gold Coast Banket Areas, Ltd	Sept. 30, 1935	153,739	0.4025	61,880				
Ashanti-Adowsena (Banket) Goldfields, Ltd	Nov. 1, 1935	85,000	0.350	29,750				
Kimingini Gold Mining Co., Ltd	June 30, 1935	45,500	0.5125	23,319				
Tati Goldfields, Ltd	Mar. 31, 1935	61,000	0.3768	22,985				
Kenya Gold Mining Syndicate, Ltd	Dec. 31, 1934	26,000	0.5569	14,480				
Kavirondo Gold Mines, Ltd	Mar. 31, 1935	30,873	0.4585	14,155				
Tanganyika Central Gold Mng. Co., Ltd	June 30, 1935	25,900	0.500	12,950				
Australia								
Mount Morgan, Ltd	June 30, 1935	7,652,530	0.2185[c]	1,629,989	June 30, 1927	8,000,000	0.2185	1,748,000
Lake View & Star, Ltd	June 30, 1935	3,035,900	0.316	959,345	June 30, 1932	1,832,437	0.400	732,974
Wiluna Gold Mines, Ltd	Aug. 31, 1935	2,040,000	0.308	628,320	Aug. 31, 1932	1,257,000	0.392	492,744
Great Boulder Proprietary Gold Mines, Ltd	Dec. 31, 1934	356,370	0.3819	136,097	Dec. 31, 1932	211,000	0.405	85,455
Sons of Gwalia, Ltd	Dec. 31, 1934	680,000[b]	0.353	240,040	Dec. 31, 1932	430,000	0.353	151,790

[a] Not including ore in shafts and pillars.
[b] Reports five years ore at present rate.
[c] Plus 1.72 per cent copper.

TABLE 3.—(*Continued*)

Company	Year Ended	Reported Ore Reserves, Tons	Average Gold Content, Oz. per Ton	Total Metallic Content of Gold, Troy Oz.	Year Ended	Reported Ore Reserves, Tons	Average Gold Content, Oz. per Ton	Total Metallic Content of Gold, Troy Oz.
Australia—(*Continued*)								
Yellowdine Gold Development, Ltd.	August 1935	130,000	1.50	195,000				
North Kalgurli (1912) Ltd.	Dec. 31, 1934	487,000	0.400	194,800	Dec. 31, 1931	274,800	0.487	133,931
Lancefield (W.A.) Gold Mine, N.L.	June 30, 1935	330,000	0.40	132,000				
New Occidental Gold Mines, Ltd.	Sept. 30, 1935	316,000	0.425	134,300				
Triton Gold Mines, N.L.	June 30, 1935	300,000	0.3125	93,750				
Golden Plateau, N.L.	June 30, 1935	232,300	0.400	92,920				
Edna May (W.A.) Amal. Gold Mines, N.L.	June 30, 1935	160,300	0.5645	90,500				
Lady Shenton Gold Mines (1934) Ltd.	July 22, 1935	57,663	1.3943	80,400				
South Kalgurli Consolidated, Ltd.	Sept. 30, 1935	259,600	0.300	77,880	Sept. 30, 1932	287,000	0.384	110,208
Norseman Gold Mines, N.L.	Oct. 16, 1935	157,000	0.475	74,575				
Marvel Loch Gold Development, Ltd.	May 31, 1935	210,000	0.250	52,500				
Golden Dyke Gold Mine, N.L.	Dec. 31, 1934	130,000	0.40	52,000				
Gold Mines of Kalgoorlie, Ltd.	Mar. 31, 1935	113,000	0.400	45,200				
Mount Magnet Gold Mines, N.L.	Dec. 1, 1935	208,918	0.209	43,664				
Tindals Gold Mines, Ltd.	Dec. 1935	100,000	0.350	35,000				
Southern Cross Gold, N.L.	Feb. 1, 1936	110,000	0.30	33,000				
Mount Coolon Gold Mines, N.L.	Dec. 31, 1934	53,000	0.560	29,680	Dec. 31, 1932	132,000	0.966	127,512
Ora Banda United Mines, Ltd.	Sept. 30, 1935	101,000	0.2675	27,018				
Rothsay Gold Mines, N.L.	July 31, 1935	57,500	0.40	23,000				
Paringa Mining & Exploration Co., Ltd.	Feb. 29, 1936	141,000	0.2	32,000				
Edjudina Gold Mining Co., Ltd	Sept. 30, 1935	200,000						
Sand Queen—Gladsome Gold Mines, N.L.	Dec. 31, 1935	42,700						
New Zealand								
The Blackwater Mines, Ltd.	Dec. 31, 1934	96,028	0.4575	43,933	Dec. 31, 1932	75,435	0.482	36,356
Martha Gold Mining Co. (Waihi) Ltd.	Dec. 31, 1934	327,300	0.397[d]	129,938	Dec. 31, 1932	322,140	0.3926	126,472
New Guinea and Fiji								
Loloma (Fiji) Gold Mines, N.L.	June 30, 1935	100,000	2.0	200,000				
Emperor Mines, Ltd.	Sept. 30, 1935	211,000	0.657	138,627				
Mount Kasi Mines, Ltd.	Dec. 31, 1934	160,000	0.55	88,000				
New Guinea Goldfields, Ltd.	Sept. 30, 1934	156,000	0.5	78,000				
Cuthbert's Misima Gold Mine, Ltd.	Apr. 30, 1935	81,340	0.3255	26,476				
Gold Mines of Papua, N.L.	Dec. 31, 1935	400,000						
Philippine Islands								
Balatoc Mining Co.	Dec. 31, 1934	1,134,883	0.4838	549,056	Dec. 31, 1932	524,496	0.714	374,490
Benguet Consolidated Mining Co.	Dec. 31, 1934	981,905	0.4376	429,682	Dec. 31, 1931	467,600	0.727	339,961
Baguio Gold Mining Co.	June 30, 1934	132,100	0.577	76,237				
Antamok Goldfields Mining Co.	Dec. 31, 1934	360,000	0.1587	57,140				
Asia								
Nundydroog Mines, Ltd.	Dec. 31, 1934	435,207	0.7295	317,484	Dec. 31, 1932	467,499	0.745	348,287
Mysore Gold Mining Co., Ltd.	Dec. 31, 1934	431,000	0.680	293,080	Dec. 31, 1932	405,000		
Champion Reef Gold Mines of India, Ltd.	Dec. 31, 1934	405,489	0.5885	238,631	Dec. 31, 1932	315,164	0.6685	210,697
Oriental Consolidated Mining Co.	Dec. 31, 1934	755,000	0.2088	157,610	Dec. 31, 1931	285,000	0.2605	74,242

Ooregum Gold Mining Co. of India, Ltd.	Dec. 31, 1934	139,400	0.469	65,379	Dec. 31, 1932	162,187		
Chosen Corp., Ltd.	Mar. 31, 1935	127,739	0.3035	38,769	Mar. 31, 1932	101,958	0.3175	32,372
Raub Australian Gold Mining Co., Ltd.	Feb. 28, 1934	36,931			Feb. 28, 1932	46,800		
America								
Empire-Star Mines Co., Ltd.	Dec. 31, 1934	400,000	0.386	154,400				
Golden Queen Mining Co.	Jan. 1935	360,000	0.4043	145,548				
Buckskin National Gold Mining Co.	Oct., 1935	67,945	0.459	31,189				
Condor Consolidated Mines, Ltd.	Mar. 12, 1935	69,480	0.4941	34,330				
Mountain Copper, Co. Ltd., Big Canyon mine	Dec. 31, 1934	240,000	0.15	36,000				
Alaska-Juneau Gold Mining Co.	Dec. 31, 1933	38,900,000						
Homestake Mining Co.	Dec. 31, 1935	15,123,573						
Como Mines Co.	1934	1,500,000			Dec. 31, 1932	15,594,058		
Stone Cabin Consolidated Mines, Inc.	June 30, 1934	600,000						
Carson Hill	Sept. 30, 1935	1,000,000	0.20	200,000				
Montezuma Apex	Aug. 31, 1935	43,000	0.22	9,460				
Canada								
Noranda Mines, Ltd.	Dec. 31, 1935	31,029,000	0.1807	5,605,993	Dec. 31, 1932	22,450,000	0.1848	4,148,790
Hollinger Consolidated Gold Mines, Ltd.	Dec. 31, 1935	7,355,318	0.3414	2,511,544	Dec. 31, 1932	6,049,548	0.3638	2,200,900
McIntyre-Porcupine Mines, Ltd.	Mar. 31, 1935	3,430,481	0.3217	1,103,583	Mar. 31, 1932	2,562,563	0.3725	960,806
Wright-Hargreaves Mines, Ltd.	Aug. 31, 1935	1,244,657	0.599	745,550	Dec. 31, 1932	951,939	0.65	623,514
Dome Mines, Ltd.	Dec. 31, 1935	2,000,000			Dec. 31, 1932	2,000,000		
Beattie Gold Mines, Ltd.	Dec. 31, 1934	4,131,300	0.1595	658,942	Dec. 31, 1933	1,197,000	0.1875	224,438
Pamour Porcupine Mines, Ltd.	Dec. 31, 1935	1,530,000	0.261	399,330				
Teck-Hughes Gold Mines, Ltd.	Sept. 1, 1935	683,709	0.4095	280,056	Aug. 31, 1932	626,489	0.61	388,158
Buffalo-Ankerite Gold Mines, Ltd.	Dec. 31, 1935	1,093,892	0.2325	254,369	Apr. 1932	80,000	0.30	24,000
Howey Gold Mines, Ltd.	Dec. 31, 1934	2,155,087	0.1	215,509	Dec. 31, 1932	1,206,150		
Pioneer Gold Mines of B.C., Ltd.	Mar. 31, 1935	307,400	0.70	215,180				
Siscoe Gold Mines, Ltd.	Dec. 31, 1935	383,217			Dec. 31, 1932	225,000		
Young-Davidson Mines, Ltd.	Aug. 31, 1935	1,600,000	0.12	192,000				
Little Long Lac Gold Mines, Ltd.	Mar. 31, 1935	225,342	0.74	166,753				
Canadian Malartic Gold Mines, Ltd.	Dec. 31, 1935	609,000	0.199	121,191				
Lamaque Gold Mines, Ltd.	Dec. 31, 1935	316,191	0.3428	108,403				
Omega Gold Mines, Ltd.	Sept. 30, 1935	560,000	0.195	109,200				
Bralorne Mines, Ltd.	Jan. 31, 1935	575,000	0.39	224,250	Dec. 31, 1932	30,000	0.774	23,222
Hedley Mascot Gold Mines, Ltd.	Jan. 1, 1935	218,000	0.48	104,640				
Ymir Yankee Girl Gold Mines, Ltd.	Oct. 24, 1935	197,900	0.41	81,139				
Pickle Crow Gold Mines, Ltd.	Dec. 31, 1934	82,000	1.05	86,100				
San Antonio Gold Mines, Ltd.	Dec. 31, 1935	226,675	0.34	77,070	Dec. 31, 1932	74,450	0.60	44,670
Columario Consolidated Gold Mines, Ltd.	June 30, 1935	75,000	0.8	60,000				
Ymir Consolidated Gold Mines, Ltd.	Mar. 18, 1935	156,790	0.356	55,930				
Sullivan Consolidated Mines, Ltd.	Sept. 30, 1935	150,000	0.357	53,571				
McKenzie Red Lake Gold Mines, Ltd.	Dec. 31, 1935	120,000	0.435	52,200				
Sheep Creek Gold Mines, Ltd.	Dec. 31, 1935	50,000	0.50	25,000				
Toburn Gold Mines, Ltd.	Dec. 31, 1935	82,650	0.57	47,111	Dec. 31, 1932	29,200	0.70	20,440
Paymaster Consolidated Mines, Ltd.	Sept. 9, 1935	166,400	0.28	46,592				
Minto Gold Mines, Ltd. (of B.C.)	Oct. 1934	100,000	0.44	44,000				
Central Patricia Gold Mines, Ltd.	Dec. 31, 1935	95,413	0.46	43,890				
Cariboo Gold Quartz Mining Co., Ltd.	Jan. 31, 1935	152,588	0.404	61,646				
McWatters Gold Mines, Ltd.	Dec. 31, 1935	78,000	0.4478	34,928				
Reno Gold Mines, Ltd.	June 30, 1935	62,479	0.624	38,987	Aug. 31, 1933	38,680	0.67	25,916
Stadacona-Rouyn Mines, Ltd.	Dec. 1, 1935	170,315	0.2275	38,746				

[d] Plus heavy silver content.
[e] Plus 709,525 tons metallic copper.

TABLE 3.—(*Continued*)

Company	Year Ended	Reported Ore Reserves, Tons	Average Gold Content, Oz. per Ton	Total Metallic Content of Gold, Troy Oz.	Year Ended	Reported Ore Reserves, Tons	Average Gold Content, Oz. per Ton	Total Metallic Content of Gold, Troy Oz.
Canada—(Continued)								
Granada Gold Mines, Ltd.	June 30, 1934	112,268	0.335	37,691				
Kirkland Lake Gold Mining Co., Ltd.	Mar. 1936	125,000						
God's Lake Gold Mines, Ltd.	Dec. 31, 1935	99,000	0.36	35,640				
Relief Arlington Mines, Ltd.	Dec. 31, 1935	97,119	0.3566	34,635				
Powell Rouyn Gold Mines, Ltd.	Dec. 31, 1934	147,945	0.233	34,471				
Gillies Lake Porcupine Mines, Ltd.	Nov. 1935	75,000	0.457	34,283				
Red Lake Gold Shore Mines, Ltd.	Dec. 31, 1935	51,850	0.66	34,221				
Premier Gold Mining Co., Ltd.	Dec. 31, 1935	133,820	0.24[d]	32,117	Dec. 31, 1932	155,467	0.31	48,195
Pontiac Rouyn Mines, Ltd.	Sept. 30, 1934	161,345	0.194	31,319				
Matachewan Consolidated Mines, Ltd.	Dec. 31, 1935	125,965	0.228	28,720				
Shawkey Gold Mining Co., Ltd.	Dec. 31, 1935	80,000	0.357	28,560				
Arntfield Gold Mines, Ltd.	Dec. 31, 1935	112,400	0.24	26,976				
Greene Stabell Mines, Ltd.	Dec. 31, 1935	41,666	0.60	25,000				
Gunnar Gold Mines, Ltd.	Dec. 31, 1935	60,000	0.4143	24,858				
Wesko Mines, Ltd.	Dec. 1, 1935	60,085	0.401	24,094				
Island Mountain Mines, Ltd.	Dec. 31, 1935	43,381	0.553	23,990				
Wendigo Mines, Ltd.	Dec. 31, 1935	60,000	0.2857	17,143				
Grange Mines, Ltd.	Dec. 31, 1934	24,000	0.8	19,200				
Tashota Goldfields, Ltd.	Mar. 30, 1935	39,450	0.45	17,753				
Argosy Gold Mines, Ltd.	Jan. 31, 1935	27,210	0.555	15,115				
McMillan Gold Mines, Ltd.	Dec. 31, 1934	59,900	0.246	14,770				
St. Anthony Gold Mines, Ltd.	Dec. 31, 1934	50,000	0.286	14,285				
J. M. Consolidated Mines, Ltd.	June 30, 1935	45,000	0.314	14,139				
Bosquet Gold Mines, Ltd.	Dec. 31, 1935	20,945	0.5315	11,132				
Central Manitoba Mines, Ltd.	Aug. 31, 1935	14,100	0.34	4,794	Apr. 30, 1932	31,800	0.402	12,784
Laguna Gold Mines, Ltd.	Dec. 31, 1935	35,000	0.5872	20,538				
Sturgeon River Gold Mines, Ltd.	Dec. 31, 1935	33,640	0.483	16,248				
Vidette Gold Mines, Ltd.	Dec. 31, 1935	15,000	0.65	9,750				
Ashley Gold Mining Corp. Ltd.	Dec. 31, 1935	10,760	0.290	3,120				
Kootenay Belle Gold Mines, Ltd.	Dec. 31, 1935	20,500						
South and Central America								
St. John del Rey Mining Co., Ltd.	Dec. 31, 1934	1,608,000[f]	0.5	804,000	Dec. 31, 1932	1,410,000[g]	0.5	705,000
New Goldfields of Venezuela, Ltd.	June 30, 1935	517,000	0.405	209,385	June 30, 1932	486,397	0.43	209,151
Frontino Gold Mines, Ltd.	June 30, 1935	453,200	0.67	303,644	June 30, 1932	220,700	0.816	180,091
Cia. Minera Agua Fria.	Dec. 31, 1934	50,000	1.25	62,500				
Botanamo Mining Corp.	Apr. 15, 1935	16,000	0.824	13,184				
Europe								
Gruvaktiebolag Bolidens.	Dec. 31, 1934	5,900,000						

[f] Plus 1,602,000 tons of low grade and 1,097,000 tons at Espirito Santo mine.
[g] Plus 1,200,000 tons of low grade.

TABLE 3.—(*Continued*)

Company	Year Ended	Reported Ore Reserves	Average Gold Content, Cents[i]	Total Metallic Content of Gold, Troy Oz.	Year Ended	Reported Ore Reserves, Tons	Average Gold Content, Oz. per Ton	Total Metallic Content of Gold, Troy Oz.
Dredging and Hydraulicking								
Bulolo Gold Dredging, Ltd.	May 31, 1935	170,000,000[h]	41½	2,017,143				
Societe des Mines d'Or de Kilo Moto	Dec. 31, 1934	Reef and Alluvial		1,768,580	Dec. 31, 1932			1,414,864
Natomas Co.	Dec. 31, 1934	290,600,000	11	913,314	Dec. 31, 1929			479,400
Yukon Cons. Gold Corp., Ltd.	Dec. 31, 1935	64,404,000	42¼	779,369				
Hixon Creek (Cariboo) Gold, Ltd.	Dec. 31, 1934	72,000,000	30	617,143				
Tano Gold Dredging, Ltd.	Jan. 8, 1935	63,117,910	31	564,187				
Amari Nigeria Tin Mines (1931) Ltd. Gold Coast Alluvials	Dec. 31, 1934	22,800,000	76	497,647				
Colombian Mining & Exploration Co. Ltd.	Mar. 31, 1935	26,762,252	49	374,672				
Viborita Gold Mines, Ltd.	Dec. 31, 1934	50,000,000	23½	338,650				
South American Gold & Platinum Co.	Dec. 31, 1934	146,819,000		373,117[j]				
Gold Coast Selection Trust, Ltd.	Sept. 30, 1935	42,000,000	27½	328,125				
Pato Consolidated Gold Dredging, Ltd.	1935	40,000,000[k]	25	285,714				
Nokomai Gold Dredging, Ltd.	Jan. 1, 1935	15,000,000	61¼	262,500				
Lupa Exploration Syndicate, Ltd.	Sept. 30, 1935	23,000,000	29½	192,625				
Rimu Gold Dredging Co., Ltd.	Dec. 31, 1934	25,000,000	26¼	187,500				
Asnazu Gold Dredging, Ltd.	Dec. 31, 1934	9,714,750	39½	109,430				
Cie. Miniere des Grand Lacs Africaines	Dec. 31, 1934			69,447				
Gold Creek Mining Co.	Feb., 1934	55,000,000	11	172,857				
Coco Grove, Inc.	Dec. 31, 1935	14,000,000	35	140,000				
Alluvial Mines								
Madam Hopkins Gold Mining Co., Ltd.	Dec. 31, 1934	792,000[m]		871,000				
Talbot Alluvials, Ltd.	Dec. 31, 1934	616,000		667,600				

[h] All figures in this group in cubic yards.
[i] Gold content in cents per cubic yard at $35 per ounce.
[j] Crude gold, plus 434,828 oz. crude platinum.
[k] Newly acquired acreage.
[m] Square fathoms.

Table 3 gives the tonnage, grade and total metallic ounces of gold in the officially reported reserves of the world's principal gold mines. All the reserves listed are from the annual reports of the companies or official statements, with but a few minor exceptions, which have come from the technical or financial press of the various countries. It is believed that all are reliable, with the possible exception of some of the new small Canadian and Australian companies, which perhaps have been a bit optimistic in calculating possible ore into actually developed tonnage. These few would have no material affect on the total. Table 3 is believed to include substantially every important property in the world and most of the lesser ones with the following exceptions: Lake Shore, in Canada, and Yuba Consolidated (dredging) in California do not report tons or yardage or metal content; the ounce content totals do not include Homestake, Alaska-Juneau or Boliden, these companies not reporting the grade of their reserves. No account can be taken in these tables of the tremendous tonnages now being developed by the 14 great Rand development companies, which doubtless will increase present Rand reserves by tens of millions of tons. Many of the properties listed did not report reserves in 1932. These were either closed down at that time, or are new properties, with insufficient development at that time to warrant an estimate of ore position.

A substantial tonnage was developed in new properties on the Rand, the Gold Coast, Australia and Canada. The paucity of legitimate information on the operations of the smaller American gold producers has probably had some bearing upon the apathy of the investing public to new American gold shares, and has made it difficult to compile any figures on them. While the United States figures are regrettably incomplete, it is doubtful that their inclusion, if available, would materially affect the total. The reserves of all these companies in the aggregate are not believed to amount to more than the reserves of some of the individual mines in South Africa or Canada.

The summary of ore reserves (Table 4) shows that the world's mines have substantially increased their reported ore reserves and total metallic content, but that the grade has dropped. This is merely the outward expression of the effect of revaluing the price of gold. At $35 much hitherto worthless material has been elevated to the classification of "ore" with the result that underground reserves—both developed and potential—have in most cases been substantially increased. The table gives the ore reserves and grades of 179 companies, ore reserves without grade for 11 companies, yardage and grade for 15 companies, metal content without yardage for one company, and square fathoms and ounces for two alluvial wash mines. A further distinction has been made by segregating the large reserves of Mount Morgan and Noranda from the Canadian totals. These two properties materially affect the total ton-

nage and grade of the other Australian and Canadian companies. While the two periods cannot be strictly compared, a rough quantitative estimate from the data available shows that revaluation has increased the tonnage of reserves by more than 75 per cent and the total gold content by about 60 per cent. Present metal reserves amount to about three years of world production or about four years of production if the output of the U. S. S. R. is not included.

TABLE 4.—*Summary of World's Reported Reserves of Gold Ore*[a]

Region	After Revaluation				Before Revaluation			
	Tonnage All Companies (Including Those Not Reporting Grade)	Tonnage (Companies Reporting Grade Only)	Total Metallic Ounces Fine Gold	Average Grade	Tonnage All Companies (Including Those Not Reporting Grade)	Tonnage (Companies Reporting Grade Only)	Total Metallic Ounces Fine Gold	Average Grade
Transvaal, South Africa	187,731,130	183,482,000	50,473,593	0.2751				
35 Rand mines	[b]	174,669,878	48,431,667	0.2773	[b]	101,034,871	35,484,737	0.3512
Rhodesian	[b]	3,315,334	1,217,558	0.3673	[b]	1,904,100	936,421	0.4918
Other African	[b]	8,922,490	3,742,264	0.4194	[b]	1,818,065	1,317,242	0.7245
Australasian	19,193,449	18,550,749	5,837,952	0.3147	[b]	12,821,812	3,745,442	
Australasian (not including Mount Morgan)	11,540,919	10,898,219	4,207,963	0.3861	[b]	4,821,812	1,997,442	0.4143
Philippines	[b]	2,608,888	1,112,115	0.4263	[b]	992,096	714,451	0.7201
Asiatic	2,330,766	2,293,835	1,110,953	0.4843	1,783,608	1,169,621	665,598	0.5691
South American	[b]	2,644,200	1,392,713	0.5267	[b]	2,117,097	1,094,242	0.5169
United States	58,303,998	2,180,425	610,927	[c]	15,594,058			
Canada	63,396,153	60,887,936	14,497,965	0.2381	37,708,286	34,277,136	8,745,833	0.2552
Canada (not including Noranda)	32,367,153	29,858,936	8,891,972	0.2978	15,258,286	11,827,136	4,597,043	0.3887
Europe	5,900,000							
Alluvial mines	[b]	1,408,000[d]	1,538,600					
Dredging and hydraulicking	[b]	1,130,217,912[e]	8,153,993	25¼¢[f]				
Dredging (companies reporting ounces but not yardage)			1,838,027				1,894,264	
World Total (by classes)								
Ores	354,346,408	284,885,857	79,996,040	0.2808		156,134,798	53,986,957	
Alluvial mines	[b]	1,408,000[d]	1,538,600					
Dredging		1,130,217,912[e]	8,153,993	25¼¢[f]				
Dredging (no yardage reported)			1,838,027				1,894,264	
World grand total			91,526,660				55,881,221	

[a] Not including Russia.
[b] All companies listed reported grade.
[c] U.S. data not comparable.
[d] Square fathoms.
[e] Cubic yards.
[f] Cents per cubic yard.

Revaluation at one stroke has uncovered countless new gold fields and has had a more profound effect than the work a huge army of prospectors could accomplish in generations with the old price still obtaining. These new gold fields are of four general classes—the former marginal and submarginal ores in existing producers, which heretofore had little or no value; the low-grade ores left in abandoned mines, which are now

capable of being mined and treated at a profit; the known low-grade deposits that never have been worked because of the impossibility of exploiting them at a profit at $20.67; and deposits sufficiently rich to be worked elsewhere but remote from transportation or in refractory ore too expensive to treat and with low recoveries.

The interpretation of statistics is fraught with peril unless actual experience tempers the conclusions. All the tonnages given should not be attributed to revaluation. Many mines of high-grade ore reported additional tonnages of high-grade ore that would be payable without the increased gold price. This is also true of many recently developed mines, although it is possible that without the stimulus of a high price these properties might have remained undeveloped. In the main, however, the general conclusion that revaluation has greatly and measurably affected the reported ore reserves is valid, and the actual effect on the total "potential" reserves, known or reasonably expected to exist, of a tonnage and grade not yet ascertainable, will be far greater in percentage than even its effect upon the reported ore reserves.

Of the various gold-mining regions it will be noted that the Rand completely overshadows the rest of the world in the matter of total reported tonnage. This is because the peculiar geological conditions obtaining there make such estimations feasible far in advance of actual withdrawal for crushing. The estimation of gold-quartz lodes and their development far in advance of needs is not as feasible, and it would appear that the actual but unmeasurable reserves of the other mines are probably of more relative importance than the reported tonnage figures would suggest.

Mining has attained a depth of 7500 ft. on the Rand—about the limit at the old price of gold. Now, however, it is possible to pay for the operation of air-conditioning equipment, and the increased working costs for keeping the workings open under the terrific rock pressure, increased hoisting charges and lost labor time. Both the Robinson Deep and East Rand Proprietary have installed American air-conditioning equipment and others will follow suit as the necessity arises. With the new price, gold mining appears possible to an ultimate depth of 10,000 ft., so that revaluation has added about 2000 ft. of depth to the working zone. From present operations and developments now in progress it would appear that the Rand can be counted on to produce over 200,000,000 oz. of gold in the future without including much of the potentially productive but totally undeveloped areas. If, however, the extreme easterly and westerly sections of the Witwatersrand prove up to recent borehole expectations, this district will appear assured of a productive life beyond the present century, and its relative importance in the scale of unmined reserves would be vastly greater than the table would indicate. The magnitude of this will be apparent when it is realized that nearly 275,000

natives and over 30,000 whites mine and crush nearly 45,000,000 tons in a year. This means stoping out about four square miles of productive reef each year. Even the greatest gold field in the world could not withstand this persistent gnawing at its vitals for many years unless its size were greatly increased by revaluation.

As with all things, there is an ultimate limit—and for gold-ore reserves it is an economic one. If costs stay reasonably near present levels reserves will have been enormously increased. If costs soar, much of the material now classified as ore will revert to unpayable ground.

Cost of Producing Gold during the Years 1934–35

As in all business enterprises, costs are the all-important item in gold-mining operations. Even a high grade of gold ore cannot be worked at a profit if the total costs of operations exceed the value of the gold produced. Conversely, almost worthless material can be mined at a profit if costs are low enough—as witness Alaska-Juneau. What can be included in ore reserves is dependent upon working costs, every reduction of costs usually increases reserves, while every rise will tend to relegate marginal ore to waste. Thus costs, by altering ore reserves, affect the lives as well as the profits of mines.

In the past half century a great deal of gold has been produced at a loss by thousands of developments and hundreds of marginal producers all over the world. The trend towards treating greater tonnages of lower grades from greater depths calls for greater capital outlays and similarly large daily expenditures. It is therefore likely that in the future the output of gold will fluctuate in more direct proportion to the profitableness of the industry than it has in the past, as large-scale operations cannot be carried on for long at a loss.

At the present price an ample volume of production seems assured, although it appears that costs will rise within the next few years—not alone from the fluctuations of the business cycle with increased labor and material costs, but also from the necessity of mining lower grade ore from greater depths and doing more development work per ton milled, all capped by greater capital charges. There is, however, ample room in the present price of gold to take care of any ordinary rise in costs. Many producers faced with rising per-ton costs will be able to keep per-ounce costs in line by raising the mill heads, but the lower grade properties will not be in such a happy position. The ace in the hole of many of the gold-mining companies against rising costs is the little recognized fact that many are treating at present a grade of ore even lower than the general tenor of their present reserves. Furthermore, taxation is bound to lessen as governmental revenues from other industries pick up.

Table 5, on production costs, includes practically every great gold mine in the world, as well as a few small ones for comparison. With very

TABLE 5.—*Production and Cost Data of the Principal Gold-mining Companies*

Company	Year	Tonnage Treated during Year	Recovery, Oz. per Ton	Gold Produced during Year, Oz.	Operating Costs per Ton before Any Charges or Taxes[a]	Production Costs per Ounce before Any Charges or Taxes[a]	Total Costs per Ounce after All Charges and Taxes[a]
SOUTH AFRICA							
Witwatersrand Field 35 Companies—Totals & Aver.	1934	40,055,750	0.2575	10,314,764	$ 4.77	$18.52	$24.84
Government G.M. Areas	Dec. 31, 1934	2,482,000	0.3690	915,851	4.36	11.82	24.92
New State Areas	Dec. 31, 1934	1,122,000	0.3928	440,678	5.01	12.77	28.82
Sub Nigel	June 30, 1935	568,200	0.7919	449,977	8.45	10.67	18.91
Springs	Dec. 31, 1934	1,077,700	0.3913	421,688	4.86	12.42	24.19
Brakpan	Dec. 31, 1934	1,479,000	0.2734	404,411	4.86	17.76	25.61
East Geduld	Dec. 31, 1934	972,500	0.3409	331,492	4.51	13.24	21.52
Daggafontein	Dec. 31, 1934	959,900	0.3375	323,925	5.28	15.65	23.17
West Springs	Dec. 31, 1934	1,104,200	0.1571	173,514	3.76	23.95	27.89
Crown	Dec. 31, 1934	3,558,000	0.2815	1,001,618	4.86	17.26	24.52
Randfontein	Dec. 31, 1934	3,791,000	0.1928	730,936	4.80	24.89	26.64
New Modder	June 30, 1935	2,175,000	0.2491	541,876	3.52	14.09	21.28
East Rand Proprietary	Dec. 31, 1934	2,102,500	0.2272	477,712	5.30	23.35	26.30
West Rand Consolidated	Dec. 31, 1934	1,430,000	0.2375	339,588	4.24	17.85	24.09
Geduld Proprietary	Dec. 31, 1934	1,116,000	0.2904	324,136	3.63	12.50	22.93
Consolidated Main Reef	June 30, 1935	1,340,100	0.2193	293,911	5.32	24.25	28.29
Robinson Deep	Dec. 31, 1934	1,200,000	0.2508	301,017	4.69	18.69	24.75
Simmer & Jack	Dec. 31, 1934	1,073,900	0.2283	245,164	5.25	22.98	25.78
Modder East	June 30, 1935	1,074,000	0.2188	235,078	4.80	21.95	28.91
City Deep	Dec. 31, 1934	1,204,000	0.1962	236,210	5.29	26.95	28.43
Van Ryn Deep	Dec. 31, 1934	944,000	0.2334	220,416	4.73	20.25	25.69
Nourse	June 30, 1935	868,300	0.2254	195,746	5.84	25.89	30.76
Langlaagte	Dec. 31, 1934	1,007,000	0.1962	197,538	4.71	24.00	26.51
Modder Deep	Dec. 31, 1934	568,500	0.3387	192,537	3.74	11.06	18.39
Modder "B"	Dec. 31, 1934	1,008,000	0.1909	192,435	3.61	18.89	23.90
Geldenhuis	Dec. 31, 1934	996,900	0.1628	162,301	4.27	26.21	29.65
Witwatersrand ("Knights")	Dec. 31, 1934	958,000	0.1584	151,793	4.68	29.51	31.91
Durban Roodepoort Deep	Dec. 31, 1934	626,000	0.2424	151,772	6.00	24.76	26.65
Rose Deep	Dec. 31, 1934	764,500	0.1800	137,583	4.98	27.66	29.14
Witwatersrand Deep	Dec. 31, 1934	590,850	0.1971	116,442	5.21	26.42	28.88
New Kleinfontein	Dec. 31, 1934	649,600	0.1751	113,707	5.47	31.24	32.04
Luipaard's Vlei	June 30, 1935	509,900	0.2192	111,801	5.08	23.19	30.11
Van Ryn Gold	June 30, 1935	689,500	0.1445	99,623	4.09	28.30	31.89
Transvaal Gold	Mar. 31, 1935	243,400	0.2285	55,603	5.02	21.97	28.35
Glyns-Lydenburg	July 31, 1935	94,700	0.3528	33,410	7.80	22.12	29.73
Nigel Gold[b]	Dec. 31, 1934	66,550	0.3112	20,711	6.82	21.92	21.92
OUTSIDE NORTH AMERICA AND SOUTH AFRICA* (Arranged by Mining Districts Within Continents)							
Africa							
Ashanti	Sept. 30, 1935	192,110	1.0388	199,650	$ 7.87	$ 7.57	$14.44
Taquah and Abosso	Mar. 31, 1935	149,687	0.3347	50,109	5.06	15.12	21.15
Ariston	Sept. 30, 1935	84,325	0.4063	34,261	8.76	21.56	24.57
Cam and Motor	June 30, 1935	308,400	0.3544	109,300[c]	4.61	13.01	20.31
Globe and Phoenix	Dec. 31, 1934	72,130	0.8299	59,863	8.94	10.77	18.60
Wanderer	June 30, 1935	204,800	0.1763	36,113	3.52	19.96	25.73

Footnotes on page 324.

TABLE 5.—(*Continued*)

Company	Year	Tonnage Treated during Year	Recovery, Oz. per Ton	Gold Produced during Year, Oz.	Operating Costs per Ton before Any Charges or Taxes[a]	Production Costs per Ounce before Any Charges or Taxes[a]	Total Costs per Ounce after All Charges and Taxes[a]
Africa—(*Continued*)							
Sherwood-Starr	June 30, 1935	78,600	0.2179	17,916[d]	5.65	$24.81	$35.52
Lonely Reef	Dec. 31, 1934	146,860[e]	0.1067	19,024[f]	3.23	27.04	29.83
Rezende	Dec. 31, 1934	77,600	0.2359	18,307	4.57	19.36	24.15
Maroc	July 31, 1935	[g]		13,661		19.67	28.04
Gabait	Jan. 31, 1935	7,517	0.559	4,202	11.82	21.14	32.21
Revue Dredging	June 30, 1935	1,260,000[h]	22½¢[i]	8,259	11¢[i]	17.03	21.77
Cie. Equatoriale des Mines	Dec. 31, 1934	[ww]	[ww]	20,055	[ww]	17.35	31.07[z]
South and Central America							
St. John del Rey	Dec. 31, 1934	240,500	0.4027	96,857	8.31	23.67	27.65
Frontino	June 30, 1935	72,040	0.6930	49,927	16.25[k]	23.45[k]	31.11
New Goldfields of Venezuela	June 30, 1935	160,135	0.3099	49,620	7.88	25.43	30.78
Botanamo	Dec. 31, 1934	23,979	0.8278	19,851	19.46	23.51	26.49
Veraguas	Dec. 31, 1934	13,275	1.1283	14,979	16.88	14.96[m]	25.85[n]
Asia							
Nundydroog	Dec. 31, 1934	218,514[p]	0.496	110,536[q]	7.18[r]	14.19	21.82
Mysore	Dec. 31, 1934	177,033	0.519	92,506[s]	9.58	18.34	24.45
Champion Reef	Dec. 31, 1934	134,770[t]	0.4621	66,400[u]	9.36[r]	19.00	24.47
Ooregum	Dec. 31, 1934	146,750[v]	0.2576	50,303[w]	6.13[x]	25.57	29.08
Oriental Consolidated	Dec. 31, 1934	183,873	0.2243	41,251	3.50	15.60	29.74
Chosen	Mar. 31, 1935	134,245	0.2855	38,323	3.00	10.51	21.69
Australasia, etc.							
Lake View and Star	June 30, 1935	524,510[y]	0.2490	132,241	5.27[x]	20.49	25.38
Wiluna	Mar. 31, 1935	439,395	0.2465	108,319	3.83	15.54	20.56
Sons of Gwalia	Dec. 31, 1934	136,548	0.3130	42,740	5.66	18.10	22.96
North Kalgurli	Dec. 31, 1934	50,383	0.4607	23,215	7.97	17.29	24.59
Blackwater	Dec. 31, 1934	31,862	0.4697	14,966	6.82	14.51	25.40
Balatoc	Dec. 31, 1934	245,128	0.5473	134,161	5.41	9.88	14.68
Benguet	Dec. 31, 1934	250,553	0.4527	113,428	6.14	13.55	16.98
Bulolo Gold Dredging	May 31, 1935	9,920,700[h]	44½¢[i]	127,901	8½¢[i]	6.66	10.14

AMERICA AND CANADA

America							
Homestake	Dec. 31, 1934	1,440,692	0.3274	471,749	$ 3.50	$10.68	$20.80
Alaska-Juneau	Dec. 31, 1934	4,302,600[bb]	0.0298	128,015	0.56	18.82[b]	21.31[dd]
Natomas	Dec. 31, 1934	18,443,253[ee]	11.27¢[ff]	59,437	4.96¢[gg]	15.42	23.75
Yuba	Feb. 28, 1935			67,096		17.77	26.95
Idaho-Maryland	Dec. 31, 1934	80,237	0.5541	43,656	8.34	15.33	21.78
Argonaut	Dec. 31, 1934	68,500	0.3117	21,357	6.69	21.45	22.75
Carson Hill	Sept. 30, 1935	234,056	0.0723	16,930	2.22	30.75	32.13
Dairy Farm	Dec. 31, 1935[aa]	59,224	0.0745	4,412	1.65	22.10	
Canada							
Hollinger	Dec. 31, 1934	1,900,490	0.2285	434,257	3.91	17.11	21.36
McIntyre-Porcupine	Mar. 31, 1935	862,100	0.281	242,236	4.19	14.91	20.82
Dome	Dec. 31, 1934	547,600	0.3723	206,163[hh]	3.88	10.30	15.39
Coniaurum	Dec. 31, 1934	138,114	0.2055	28,390	5.54	26.97	35.58
Buffalo-Ankerite	Dec. 31, 1934	131,718	0.1557	20,503	3.64	23.34	30.51
Anglo-Huronian "Vipond"	July 31, 1935	104,764	0.1276	13,367	4.62[ii]	36.22[ii]	41.79[jj]
Lake Shore	June 30, 1935	833,094	0.5526	460,442	5.50	9.95	17.25
Wright-Hargreaves	Aug. 31, 1935	350,196	0.6044	211,674	6.37	10.54	17.03
Teck-Hughes	Aug. 31, 1935	383,958[kk]	0.3640	144,384[mm]	5.62[nn]	14.95	19.48
Sylvanite	Mar. 31, 1935	124,956	0.405	50,601	5.93	14.65	24.30
Macassa	Mar. 31, 1935	66,534	0.4545	30,260	7.45	16.39	22.28

[ww] Hydraulicking. [z] Francs at 6.5688 cents.

TABLE 5.—(*Continued*)

Company	Year	Tonnage Treated during Year	Recovery, Oz. per Ton	Gold Produced during Year, Oz.	Operating Costs per Ton before Any Charges or Taxes[a]	Production Costs per Ounce before Any Charges or Taxes[a]	Total Costs per Ounce after All Charges and Taxes[a]
Canada—(*Continued*)							
Toburn	Dec. 31, 1934	36,230	0.563	20,401	9.89	17.57	28.27
Kirkland Lake Gold	Dec. 31, 1934	64,952	0.3128	20,316	7.63	24.39	27.19
Barry-Hollinger	Dec. 31, 1934	33,445	0.1318	4,409	6.00	45.49	
Howey	Dec. 31, 1934	396,109[pp]	0.1160	45,985	2.15	18.49	25.11
Ashley	Dec. 31, 1934	43,532	0.3028	13,181	8.69	28.68	35.72
Central Patricia	Dec. 31, 1934	11,536[qq]	0.5524	6,373	11.28	20.43	28.03
Parkhill	Sept. 30, 1935	20,100	0.4622	9,290	13.39	28.97	32.67
San Antonio	Dec. 31, 1934	64,294	0.3365	21,638	5.49	16.30	23.08
Central Manitoba	Aug. 31, 1935	56,068[rr]	0.2661	14,922	8.67	32.59	42.16
Siscoe	Dec. 31, 1934	124,151	0.5106	63,394	5.56	10.89	15.27
Beattie	Dec. 31, 1934	359,200	0.1473	52,905	3.72	25.23	31.57
Granada	Dec. 31, 1934	35,424	0.1937	6,864	7.46	38.48	47.55
Pioneer	Mar. 31, 1935	130,545	0.6646	87,030[ss]	5.42	8.14	18.38
Bralorne	Dec. 31, 1934	98,664	0.4591	45,296	4.44[tt]	9.67[tt]	18.39
Premier	Dec. 31, 1934	154,693	0.248[uu]	38,371	4.52		[vv]
Reno	June 30, 1935	33,943	0.6050	20,538	9.95	16.44	22.74
Cariboo	Dec. 31, 1934	28,772	0.3828	11,014	9.68	25.28	32.42

* All conversions made with Sterling at $4.90; Australian pounds at $3.90 and Canadian dollars at par.

Costs per ton are for mined ore except where noted. Costs per ounce are based on total gold output including recoveries from tailing treatment, if any. The column on recovery refers to mined ore, and is not necessarily based on gross output in ounces divided by total tonnage milled.

Operating costs include usual mining and treatment charges and current development expense. In addition refining, mint and marketing charges and local administration costs and general charges, where it is possible to segregate items.

Total costs include head-office expenses, where possible to segregate, depreciation, depletion, when charged, directors regular fees, but not extra compensation or bonuses; and all forms of taxes except taxes on dividends. In the various countries there are income, royalty, government lease participation, gold export duties, excess profits and other taxes. Where gold is required to be sold to local governments at a price below the world market, this constitutes in effect a tax on production and has been added to the costs per ounce.

[a] All costs based on Sterling at $4.90. The first eight companies have properties leased from the Government and subject to royalties and in some cases additional taxes.

[b] From start in July, 1934. Leased but no lease payment, taxes or phthisis charges during first year.

[c] Includes 2582 oz. from slags.

[d] Includes 782 oz. from slags.

[e] Plus 66,400 tons of tailings.

[f] Includes 3355 oz. from tailings treatment.

[g] Production mostly alluvial, some lode.

[h] Cubic yards.

[i] Per cubic yard with gold at $35.

[j] Per cubic yard.

[k] Includes depreciation.

[aa] Two year period ending Dec. 31, 1935.

[bb] Tons trammed to mill.

[cc] $17.90 crediting silver and lead.

[dd] $20.39 crediting silver and lead.

[ee] Cubic yards.

[ff] Per cubic yard with gold at $35.

[gg] Per cubic yard.

[hh] Includes 2266 oz. from retreatment.

[ii] Mine expense only; no general or administrative expense.

[jj] Before taxes.

[kk] Plus 40,290 tons of tailings.

[m] $13.26 after crediting copper and silver.

[n] $24.14 after crediting copper and silver.

[p] Plus 29,254 tons of tailings.

[q] Includes 2138 oz. from tailings treatment.

[r] Including a proportionate amount of tailings

[s] Includes 612 oz. from slags.

[t] Plus 159,109 tons of tailings.

[u] Includes 4117 oz. from tailings treatment.

[v] Plus 249,250 tons of tailings.

[w] Includes 12,358 oz. from tailings treatment.

[x] Mined ore only.

[y] Including 14,728 tons of tributer's ore, but not 22,480 tons of tailings. The metal from the latter, however, is included in ounce output and cost figures.

[mm] Includes 4610 oz. from tailing treatment.

[nn] Includes proportionate tailings; mine ore alone estimated at $5.17.

[pp] 481,757 tons mined; 85,648 sorted out.

[qq] From start of operations May 27, 1934.

[rr] Period 16 months to Aug. 31, 1935.

[ss] Includes 267 oz. from slags.

[tt] Exclusive of development cost.

[uu] Includes 4.246 oz. silver.

[vv] Extremely high silver makes cost after crediting silver content not comparable.

few exceptions these were calculated from the annual reports of the companies; the others are from figures given by foreign financial papers and services. All the figures are for the latest fiscal period for which reports have been made public.

In surveying the costs of the world's great gold mines we find a great variation in the costs per ton and per ounce at the various properties, even in the same gold field. In 1913 on the Rand working costs averaged $13.30 per ounce, and elsewhere in the world ranged from $7.00 to more than the Mint price of $20.67. In 1920 costs on the Rand had risen to an average of $18.99 per ounce, owing to the general dislocation caused by the war; by 1925 these costs came down to $14.23 (only 8¢ more than the record low) and in 1929 were $14.64 per ounce. During the past year of record (1934) costs rose to $18.52 per ounce, owing mainly to handling larger tonnages of lower grade ore, but the costs per ton were actually 2¢ less than 1929. Therefore it can be seen that a rise in per-ounce costs can take place while per-ton costs are dropping.

Much is made of the low costs per ton achieved by certain gold mines. This information is of definite value and use for the engineer in studying methods, relative costs and possible savings, but it is of only academic interest to the shareholder. The ultimate determining factor of the success of any gold-mining enterprise is the cost per ounce—not of working the ore but the cost after every charge has been added in; the difference is what he may expect in some measure to be returned to him for his investment.

The study of comparative costs involves a great number of diverse factors, and to secure any value from the result all these factors must be taken into consideration because comparisons made blindly can only lead to error. At best, such comparative cost studies are not completely satisfactory and never can be made so because of the impossibility of reducing all components to a common index.

In the accompanying tables it has been deemed advisable to include for each property not only the tonnage, but the grade of the ore and the total ounces produced during the period. Appropriate footnotes have been made where tailings are being treated and whether such treatment costs are included in the total operating costs per ton. Other properties give the costs per ton, without development and it is not possible to arrive at a uniform basis for costing. These exceptions have been noted. In the space available it was impossible to enter many other details of operations that would have a direct bearing on the relative costs, and in most cases these details are not available. The character of the deposit, the width of ore mined, its hardness, the depth of workings, timbering problems, water-pumping costs, ventilation, relative ease of extraction of the metallic contents, accessibility of the region for labor and transport of supplies, efficiency of labor, climate, government mining

regulations, taxation, as well as a host of other variables that have their effect directly upon working costs and indirectly on other charges. These factors have to be appraised in making comparisons, but such details are impossible to enter upon a comparative table, and can only be used with proper judgment by experienced persons. By arranging the companies by districts, some of these variables are eliminated, but there still exist a myriad of differences between two properties even in the same camp. For the purposes of this survey, however, most of these factors can be disregarded and attention focused on the costs per ton and per ounce of production and the wide spread between these costs and the actual over-all costs at most of the mines. The operating costs and charges have been segregated where possible from nonoperating costs, and in general an attempt has been made to place accurately the various items. The over-all costs include depreciation, depletion where charged, silicosis assessments, taxes in all forms except taxes on dividends that were considered in effect a reduction of the dividend payable to the shareholder. When the mine had to sell its gold to a local government for less than the world price obtaining during that period, this difference was considered as a form of taxation and added to the total costs. For the leased South African mines, lease royalties were considered not as an operating charge but as an overhead, and the costs of the Indian mines given in their reports were altered and royalties taken out of working costs and put in over-all costs.

All the large Rand properties are included and space does not permit mentioning these properties in detail. Sub Nigel is the richest of the producers and treats one of the smallest tonnages. Its per-ton costs are high because it mines a relatively narrow reef and rejects more rock than it mills. Notwithstanding this its per-ounce costs are extremely low. Randfontein treats the largest tonnage and handles one of the lowest grade ores.

Of the very large mines, Ashanti treats the richest ore—its per-ounce costs being lower than its per-ton costs; a striking difference from Alaska-Juneau, which treats the lowest grade of gold ore in the world. In all the company costs it will be noted that the per-ton costs taken as a whole have shown striking stability during the past few years, but that, because leaner ores are treated, the working costs per ounce have risen to a modest extent. The over-all costs, however, have mounted materially, showing the alacrity with which hard-pressed governments have assessed taxation to relieve the gold miner of a goodly proportion of the gain caused by the increase in the world price for gold.

Recent Trends in Copper Production, Ore Reserves and Costs

By John J. Croston,* Member A.I.M.E.

(New York Meeting, February, 1937)

In the closing months of 1936 the copper industry gave every evidence that it was at last on the threshold of an improved era. At the beginning of the year prices stood at 9¼¢, which in itself was a great improvement over prices prevailing for the three previous years. Industry generally was showing signs of moving forward in this country, and business was definitely good in certain foreign quarters. The threat of war and preparations for rearmament in Europe and Asia acted as a further stimulus to purchasing. By midsummer the situation had so improved and stocks had been so reduced that the cartel of foreign producers (including American producers with foreign mines) decided to increase the production quota to 75 per cent on Aug. 1. Large-scale buying gave continued strength to the price and by Oct. 1 the quota was increased to 80 per cent, and again on Oct. 15, to 85 per cent. Notwithstanding this the price of copper continued its advance and on Nov. 1, with stocks declining continuously, the cartel increased operations to 95 per cent, then to 105 per cent on Nov. 5. In the meantime the domestic producers, because of the great increase in home consumption, had increased their production by nearly one-half.

At the close of the year copper sold at 12¢ per pound, and since the turn of the present year has reached 17¢. This rapid increase has not been wholly justified. Much of the increase was due to the legitimate demands of recovering industry, but rearmament orders plus accumulation of reserve stocks in case of war, purchases in anticipation of labor strikes and purely speculative activities on the London Metal Exchange accounted for the balance. Certainly in view of the state of industry and the general world price level, it appears unnecessarily high. Talk of a copper shortage is sheer nonsense, the recent condition in the market being created by overbuying, leaving but a small volume for speculative supplies. It takes some months for an increase in mine output to reach the market, but production is again overtaking apparent consumption, stocks have begun to increase, and this, together with the reopening of many small

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producers, probably will act as a brake on further price increases. High copper prices have always served to spur the production from marginal mines, and also to bring much secondary copper into the market.

World production in 1936 was the highest since 1929, at approximately 1,800,000 short tons compared with 2,118,000 tons the former year. This is an increase of about 200,000 tons over 1935, when 1,603,000 tons was produced, and is far above the low of 987,000 tons registered in 1932. The latter year showed the lowest production in a quarter of a century, if we except the postwar years 1921–1922, when the world was glutted with wartime copper stocks.

Apparent consumption was approximately 2,000,000 tons, reducing stocks by some 200,000 tons. However, the character of some of the purchasing leads to the belief that a considerable amount of this copper is not going into actual consumption but was purchased for speculative purposes or to be held as a war reserve, and should the war clouds clear it may find its way back into the market to depress prices later on.

Production Trends

In the years preceding the Civil War, and up to 1869, the principal individual copper producer of the world was the Mansfeld mine, which has produced more or less regularly since the twelfth century. In that year Calumet & Hecla forged ahead, although its production was less than 6200 tons of copper. By 1877 the reorganized mines of Rio Tinto took premier position, with an output of slightly more than 27,000 tons, and maintained leadership until displaced by Anaconda in 1892 with an output of 37,500 tons. The first mine to produce more than 50,000 tons a year was Anaconda, in 1896, and with the exception of the years 1905–1907, when Calumet & Hecla again led, and 1908–1909, when the Copper Queen led, it maintained its position as the world's greatest copper mine until just before the depression.

A study of the producers of a quarter of a century ago reveals that there were about 150 mines producing copper in substantial quantities, but only 26 had outputs in excess of 10,000 tons of metal annually. Of these only 10 produced more than 20,000 tons and but two more than 50,000 tons, while there was but one mine capable of turning out 100,000 tons a year. The beginning of work on the porphyries brought in an era of large-scale low-cost mining, and the introduction of the flotation process and leaching made possible reasonably high recoveries at low cost. Today, through mergers, consolidations and integration of the industry, a half dozen large units control the destinies of the copper trade of the world, with another half dozen smaller units sharing most of the remainder.

These companies, several of which can each produce about 500,000 tons of metal annually, have garnered the choicest ore reserves of the

world and will continue to dominate world production without serious competition for many years to come. The smaller companies treating richer ores but with higher costs and slender reserves will have to operate under the umbrella of the giants. Improved mining methods, flotation, leaching and other processes have been a much greater boon to the large low-grade producers than to the smaller and richer mines. Usually the orebodies of these smaller producers are not susceptible to the economies of such mining methods, and the improvements in recoveries and lowering of costs of newer treatment processes are either not advantageously employable or exert but a minor effect on production costs.

Having the technical skill and the necessary finances, it appears probable that any new deposits of size will gravitate to the control of the present great producers, as the funds required for the large-scale development and equipment of a great copper deposit are prodigious. The copper industry has completed the same cycle observed in other great industries—the concentration of the business into fewer hands. There will always be a considerable number of small copper producers, but they will no longer have any considerable weight in the industry.

The period since the World War has seen the rise of the British as important factors in world copper production, and today the streams of copper flow from mine to market quite differently from the way they did 10 years ago. At that time American producers dominated world markets, American-owned companies controlled most of the Latin-American production and in addition refined most of the rest of the world's copper. Now Katanga refines its own production in Belgium, the Rhodesian output goes to England for treatment, Canadian production is refined within the Dominion, while other Continental refineries are handling business formerly done here. Even the Japanese are treating some of the Chilean output as well as their own, and recently have contracted to handle the output of the Granby concentrator.

Within the past few years a tariff wall has been erected for the protection of domestic producers. America no longer consumes more copper than all of Europe, although she may do so again later, for Europe has been using more and more copper per capita for some years. Should the trend continue, European-controlled mines will share in a large part of this business. Domestic producers apparently will operate primarily to supply domestic demands, and American-controlled foreign producers will sell in foreign markets in competition with European-controlled companies, or, when prices are high enough and demand sufficient, will supplement the needs of the domestic fabricators.

The next decade will witness the inauguration of several new large producers. One company, N'Changa Consolidated Copper Mines, Ltd., has just been organized with a capital of about $25,000,000 to exploit some of the ore bodies owned by Rhokana. Others are to be anticipated

in Northern Rhodesia, Belgian Congo and Uganda, all controlled by existing copper interests. There is also the possibility of copper from African colonies of France and Portugal if developments are favorable. In Latin America Anaconda has the Santiago property in reserve, and there are the Rio Blanco and Ferrobamba deposits. A number of properties are under development in Sweden, Finland, Serbia, Turkey and elsewhere in Europe, which might in the aggregate turn out substantial tonnages of copper. In Canada a number of properties are nearly ready for production, on some plants have been built but not yet operated on account of the condition of the market, but soon Sherritt-Gordon, Waite-Amulet, Aldermac, Normetal and perhaps Coast Copper will be adding their quota. Here in the United States, the Mountain City property of Anaconda is already in production. Howe Sound is building a concentrator for its Chelan property, while Phelps Dodge has a large tonnage available at Morenci when times are propitious. Bagdad may get into moderate sized production in time. These potential producers of the future will in the aggregate, together with the expected increases in output of large existing producers, more than offset any decline in output through the exhaustion of older properties, and serve to assure the world of adequate supplies of copper for a long time to come. While a considerable increase in the total consumption of copper is to be expected with the passing years, the rate of increase of production of virgin metal should gradually taper, and more of the demand be filled by secondary copper. It is probable that secondary copper, important as it is now, will play a vastly greater role in the future.

Table 1 gives the output of the world's principal copper mines. The figures were assembled from annual reports of the individual companies, official statements or private communications, except where noted as unofficial estimates, calculations from copper content of ores, concentrates or matte, etc. The latest available data cover the year 1935, and comparison is made with 1932, the low point of recent copper production. The years 1929 and 1930 cover the culmination of the recent boom, while 1912 enables us to look back a quarter of a century. The present dominance in the industry of the newer producers will be noted. Not included in the table are a large number of copper mines that were in production in 1912, but have since shut down permanently.

Present Copper Ore Reserves

While the years of the depression have not been conducive to the expenditure of funds for the exploration of new copper reserves, there have been a few notable exceptions. These, together with information recently made available on properties hitherto not included in any tabulation, have made it appear desirable to make a new appraisal of the known underground reserves. Several tabulations have appeared in the

TABLE 1.—*Annual Output of World's Leading Copper Mines*[a]

SHORT TONS

Company	1935	1932	1930	1929	1912[b]
Custom Smelters:					
Amer. Smelt. & Ref. Co	204,452	138,648	440,784	616,398	66,000
Amer. Metal Co	129,061	107,074	235,666	245,856	
U. S. Smelt., Ref. & Min. Co.	1,671	531	2,986	2,558	10,576
Groups:					
Anaconda[c]	258,972	139,339	323,706	495,285	173,117
Kennecott[d]	209,135	82,536	173,058	250,567	130,000
Phelps-Dodge[e]	88,438	41,544	56,479	88,590	141,723
Mines:					
Chile[f]	131,994	40,685	89,596	149,788	[i]
Katanga	118,698	59,595	153,165	151,008	2,664
International Nickel[g]	116,505	28,832	54,872	40,917	11,000
Braden[h]	112,010	49,871	80,993	88,163	4,750
Anaconda (Butte)[f]	77,371	48,787	98,617	148,507	148,487
Utah[h]	59,233	30,006	80,569	148,313	45,683
Roan Antelope[i]	56,753	42,233	1,133	[i]	[i]
Rhokana[i]	56,447	54,408	[i]	[i]	[i]
Bor	42,990	33,245	26,966	22,790	8,250
Noranda	37,239	31,507	38,071	25,813	[i]
Nevada Consolidated[h]	33,830[k]	29,992	70,990	133,137	62,757
Calument & Hecla & subsidiaries	33,336	16,899	58,199	67,347	65,006
Cerro de Pasco	31,989	22,910	43,200	49,993	28,219
Mufulira[i]	31,498	[i]	[i]	[i]	[i]
Andes[f]	28,641	11,619	47,023	81,332	[i]
Rio Tinto	[l]	[l]	[l]	[l]	44,716
Mansfeld	25,890	27,703	23,268	25,235	22,600
Hudson Bay	24,509	21,079	1,186	[i]	[i]
Greene-Cananea[f]	21,061	18,410	21,212	29,413	24,630
Nippon[m]	[n]	19,518	24,865	19,225	[n]
Furukawa	19,010	20,035	18,536	17,886	14,763
Mount Lyell[o]	15,642	12,271	10,995	8,739	5,159
Magma	15,193	10,853	15,942	19,128	[i]
Miami	14,870	7,907	34,600	29,421	16,416
Mitsubishi	14,672	15,207	14,040	12,092	11,287
Sumitomo	14,475	14,005	18,895	20,792	10,024
Outokumpu	12,180[p]	7,329	5,680	4,960	[i]
Granby	11,752	19,324	23,416	30,427	14,553
Fujita[q]	[i]	10,370	10,954	10,139	8,795
Cyprus[p]	13,000	2,305	4,500	5,000	[i]
Northern Peru Min. & Smelt. Co.[r]	[n]	[n]	10,701	11,158	[i]
Messina[i]	10,656	10,598	9,751	7,616	700
United Verde Extension	9,985	17,876	22,783	32,056	[i]
Tennessee[s]	8,964	3,815	7,190	7,624	6,626
Boleo	8,670	11,487	13,889	12,903	14,168
Copper Range	8,380	6,094	11,897	12,099	38,909
Indian Copper	7,728	4,962	3,331	1,831	2,683[t]
Matahambre	7,672	6,533	17,299	16,515	2,500[u]
Howe Sound[g]	7,214	1,103	22,633	21,516	7,150
M'Zaita (Chagres)	6,704	[n]	4,100	3,000	3,480
Naltagua	5,926	5,460	5,339	6,770	1,225
Sulitjelma	5 811	[l]	[l]	[l]	5,500
Bolidens	5,785	3,805	[i]	[i]	[i]
Tocopilla	5,264[v]	[n]	5,207[v]	[n]	250
Disputada de Las Condes	5,016[v]	[n]	3,990[v]	[n]	[n]
Orkla	[l]	[l]	[l]	[l]	[l]
Mountain City	4,100	[i]	[i]	[i]	[i]
Tharsis	[l]	[v]	[l]	[l]	3,782
Burma[i,w]	3,946	5,871	8,416	[n]	[n]
Inspiration	3,758	8,512	32,803	53,654	[i]
Mother Lode	3,333	1,716	4,823	6,121	[i]
Buchans[r]	3,259	2,373	1,054	1,032	[i]
Cons. Copper & Sulphur	2,339	3,446	2,489	2,177	925
Falconbridge	2,515	1,197	656	[i]	[i]
Mt. Morgan[i]	1,668	259	[i]	[i]	10,394[x]
Shattuck-Denn	1,381	[y]	4,167	6,368	1,255
Minor producers during 1935:					
Walker	822	535	7,888	7,516	[i]
Huelva Copper[i]	[n]	1,534	[n]	2,281	[n]
Pena Copper	343	645	1,008	923	888
Sunshine	673	357	198	133	[i]
Carlota	635	[n]	[n]	[n]	[n]
Shenandoah-Dives	367	879	629	315	[i]
Ohio	380	[v]	1,024	1,108	3,277
Cons. Min. & Smelt. of Canada[g]	319	384	7,064	4,173	1,457[o]
Roros	211	[l]	[l]	[l]	650
Mason & Barry	[n]	257	226	218	3,963
Ouancos	[l]	[l]	[l]	[l]	[l]
Majden Pek	[l]	[l]	[l]	[l]	[l]

TABLE 1.—(*Continued*)

Company	1935	1932	1930	1929	1912[b]
Minor producers during 1935 (*cont.*):					
Plakalnitza		[l]		[l]	
Bagdad	86[z]	[j]	[j]	[j]	[j]
Important producers idle during 1935:					
Consolidated Coppermines[y]		870	16,306	11,366	1,919
Otavi Mines[y,aa]		2,646	16,645	13,889	6,435
Quincy[y]		[y]	5,470	2,230	10,317
Sherritt-Gordon[y]		4,965	[j]	[j]	[j]
Mazapil[y]		[y]	4,378	6,239	6,778
California-Engels[y]		[y]	2,036	5,550	[j]
Corocoro United[y]		[y]	1,654	4,528	2,058
Poderosa[y]		577	2,711	2,905	2,539
Iron Cap (Christmas)[y]		107	3,462	2,209	[j]
Tezuitlan[i,v]		[y]	1,781	1,719	[bb]
Gatico		[y]	[n]	[n]	1,936
Mindouli		[y]	198	639	[j]
Equipped for production:					
Waite-Amulet	[y]	[y]	[j]	[j]	[j]
Aldermac	[y]	[z]	[y]	[j]	[j]

[a] Includes principal groups and custom smelters. Does not include U.S.S.R.
[b] Figures for 1912 include production of individual companies since consolidated.
[c] Own mines and subsidiaries (Chile, Andes, Greene-Cananea). Custom ores (including Walker and Mountain City) and toll treatments.
[d] Own mines in Alaska, Utah, Nevada (Arizona and New Mexico shut down since April 1933 and October 1934, respectively) and Chile.
[e] Properties in United States only since September 1931 (when Moctezuma in Mexico was shut down) including Calumet & Arizona and New Cornelia, but not United Verde, recently acquired.
[f] Included in Anaconda total.
[g] Sales, not production.
[h] Included in Kennecott total.
[i] Year ended June 30 following.
[j] Not yet in operation.
[k] Ely alone, previous years include Ray and Chino.
[l] No recent figures available—approximate relative rank shown by position.
[m] Saganoseki, Hidachi and smaller properties; in 1912 Hidachi alone produced 8651 tons.
[n] Not available.
[o] Year ended September 30.
[p] Unofficial estimate.
[q] Kosaka only.
[r] Included in American Smelting & Refining Company total.
[s] Not including Ducktown, since acquired.
[t] Old Cordoba output.
[u] Estimate under old ownership.
[v] Estimated copper content of concentrate.
[w] Estimated copper content of matte.
[x] Year ended May 31 following for predecessor
[y] Shut down.
[z] Small experimental production.
[aa] Year ended March 31 following.
[bb] Shut down during 1912.

past, notably those of the American Bureau of Metal Statistics, International Geological Congress and P. E. Barbour. Messrs. Riddell and Jermain have recently estimated Russian reserves, members of the International Geological Congress have estimated the reserves of certain European mines, and there have been changes in other countries.

It was believed desirable to include the copper reserves of mines other than copper mines, inasmuch as this metal will find its way into the market during the exploitation of the other metals, and it is to be considered just as much a copper reserve as the metal contained in the "straight" coppers. From a price standpoint, this metal should be accorded even greater importance because it will come into the market regardless of the price and state of the copper market should the receipts from the sales of the other metals be sufficiently attractive. If there were many more mines like International Nickel, Noranda, Hudson Bay,

TABLE 2.—*World Copper Reserves*

INCLUDING COPPER CONTAINED IN IMPORTANT TONNAGES IN OTHER METAL DEPOSITS

Company	Date of Estimate	Reported Ore Reserves, Short Tons	Average Copper Content, Per Cent	Total Metallic Content of Deposit, Short Tons of Copper
United States:				
Anaconda (Butte)	Oct. 1, 1935	75,000,000[a]	4.00	3,000,000
Bagdad Copper	Dec. 31, 1932	48,000,000	1.08	518,400
Cons. Coppermines	Apr. 20, 1934	35,000,000	1.10	385,000
Gray Eagle		1,045,000	3.23	33,754
Inspiration Cons	Oct. 1, 1935	69,010,770	1.373	947,518
Miami Copper	Jan. 1, 1936	84,624,869	0.938	794,163
Mother Lode	May 26, 1934	104,000	11.59	12,054
Mountain City	Oct. 1, 1935[b]			100,000[a]
Kennecott Copper[c]				
Ely property	July 5, 1935	77,000,000	1.22	939,400
Ray property	July 5, 1935	81,448,000	1.51	1,229,865
Chino property	July 5, 1935	170,839,000	1.23	2,101,320
Bingham property	Dec. 3, 1930	640,000,000	1.07	6,848,000
National Copper	1929	900,000	2.3	20,700
Old Dominion	1931	2,000,000	2.00	40,000
Phelps Dodge[d]	1932	388,146,550	1.13	4,386,056
Seneca Copper	Dec. 31, 1930	3,486,895		
Tennessee[e]	Dec. 31, 1935	6,148,132	1.60	98,370
United Verde Extension	Dec. 31, 1935	61,000	7.00	4,270
Van Dyke	Apr. 31, 1931	1,000,000	5.00	50,000
Not reported. Unofficial estimates:				
Michigan	1934	100,000,000	1.00	1,000,000
California, Montana, New Mexico, Arizona, Tennessee, etc.		70,000,000	2.143	1,500,000
Canada and Newfoundland:[f]				
Aldermac	Apr. 1, 1936	1,743,760	2.00	34,875[g]
British Columbia Nickel	Feb. 28, 1936	1,042,200	0.46[h]	4,794
Denison Nickel	1936	741,240	1.05[i]	7,783
Falconbridge Nickel	Dec. 31, 1935	4,059,475	0.91[j]	36,941
Granby Cons	Dec. 31, 1933	13,449,900	1.81	243,443
Great Falls	Dec. 31, 1935	300,000	1.00[k]	3,000
Howe Sound (Britannia)	Dec. 31, 1935	10,000,000	1.30	130,000
Hudson Bay	Dec. 31, 1935	24,770,000	2.10[l]	520,170
International Nickel	Dec. 31, 1935	205,590,592	2.00[m]	4,111,812
Mandy	1929	98,000	6.5[n]	6,370
Noranda	Dec. 31, 1935	31,029,000	2.51	779,082[o]
Normetal	Jan. 1, 1934	700,000	3.00[p]	21,000
Ontario Nickel	Apr. 1, 1936	116,000	1.04[q]	1,206
Sherritt-Gordon	Dec. 31, 1932	4,783,175	2.41[r]	115,419
Sudbury Basin	Dec. 31, 1929[s]	833,000	2.80[t]	23,533
Sunloch	1918[s]	4,500	3.00	4,500
Waite-Amulet	Dec. 31, 1933	955,445[u]	4.50	43,023
Buchans	Dec. 31, 1934	6,375,000	1.50[v]	94,625
Newfoundland:				
Gull Lake	1929	1,300,000	2.70	35,100
Mexico:[w]				
Greene-Cananea	Oct. 1, 1935			240,000
Moctezuma[y]	1930	3,500,000	2.70	94,500
Cuba:				
S.A. Minas de Matahambre	1933	939,000	4.75	44,603
Nicaragua:				
Tonopah Nicaragua	1918[s]	1,492,088	5.059[z]	75,485
Venezuela:				
South American	1932	400,000	3.50	140,000
Peru:[aa]				
Fundicion de Magistral		3,000,000	7.00	210,000
Ferrobamba		6,778,000[bb]	3.56	241,450
Chile:[cc]				
Andes Copper	Dec. 31, 1935	112,549,287	1.47	1,654,475
Kennecott (Braden)	Dec. 31, 1935	205,019,050	2.18	4,469,415
Chile	Dec. 31, 1935	938,931,000	2.15	20,187,023
Rio Blanco[dd]		42,000,000	2.3	966,000
Santiago[ee]	Dec. 31, 1921	52,819,282	2.11	1,116,078
Australasia:				
Lake George	1931	2,880,000	0.75[ff]	21,600
Mount Elliott	1929	1,916,550		100,936
Mount Lyell	Sept. 30, 1936	8,825,000	1.90[gg]	167,675
Mount Morgan	June 30, 1936	7,506,652	1.77[hh]	132,868
Wallaroo and Moonta	1935	618,000	3.77	23,302
Asia:[ii]				
Burma	June 30, 1936	3,914,182	0.87[jj]	34,053
Indian	Dec. 31, 1935	950,801	3.19	30,331

TABLE 2.—(*Continued*)

Company	Date of Estimate	Reported Ore Reserves, Short Tons	Average Copper Content, Per Cent	Total Metallic Content of Deposit, Short Tons of Copper
Europe:[kk]				
Huelva Copper	[mm] 1933	3,358,739	1.10	36,946
Huelva (Pyrites de)	[mm] 1933	6,613,860	0.58	38,360
Imperial Chemical	[mm] 1933	3,913,201	1.04	40,593
Pena	[mm] 1933	3,314,634		
Rio Tinto	[mm] 1933	5,000,000	1.40	70,000
St. Gobain	[mm] 1933	1,653,465	1.39	22,983
San Platon	[mm] 1933	440,924	3.50	19,290
Seville	[mm] 1933	5,511,550		
Tharsis	[mm] 1933	100,861,365	0.75	756,460
Bolidens Gruvaktiebolag	1934	5,900,000	2.00	118,000
Outokumpu	[nn] 1933	9,369,635	4.00	374,785
International Nickel-Petsamo	[nn] 1934	5,511,550	1.30	71,650
Foldal	[oo] 1933	330,693	1.63	5,401
Grong	[oo] 1933	8,487,787	2.21	187,392
Orkla	1935	20,000,000	2.5	500,000
Roros	[oo] 1933	771,617	1.00	7,716
Sulitjelma	[oo] 1933	5,070,911	1.77	89,622
Mansfeld A.G.	[pp] 1933	[qq]		650,163
Rammelsberg	[pp] 1933	1,615,986	4.27	69,005
Stadtberger Kupferhutte	[pp] 1933	688,937	1.6	11,023
Sontrau-Richelsdorf	1936	[rr]		440,924
Plakalnitza	[ss] 1933	165,347	2.50	4,134
Cyprus		20,000,000[tt]	2.10	420,000
U.S.S.R.:[uu]				
Developed or drilled				6,302,347
Probable				4,146,449
Possible				7,421,412
Total		1,532,398,293		17,879,468
Africa:[vv]				
Katanga	Dec. 31, 1934	78,000,000	6.50	5,070,000
Sud Katanga	1934			
Tchinsenda Mine		16,534,650	4.00	661,386[ww]
Mokambo Mine				338,614
Mufulira	June 30, 1936			
All properties		160,390,000	4.12	6,608,068
Rhodesia-Katanga	Aug. 1, 1936			
Kansanshi Mine		11,000,000	4.25	467,379[xx]
Rhokana:				
All properties	June 30, 1936	269,454,435	4.12	11,101,523
Roan Antelope	June 30, 1936	95,637,987	3.43	3,280,383
Falcon Mines	1933	56,000	7.3	4,088
Messina (Transvaal)	June 30, 1936	1,268,030	2.231	28,290
Zambesia:				
Kolembe Mine	1932	330,000	4.50	14,850

[a] Boston News Bureau, Dec. 16, 1935: "The engineering expert who examined the Montana properties for the bankers gave it as his opinion that the Butte mines can produce 300,000,000 lb. of copper annually for the next 20 years." The estimate of 75,000,000 tons is based on the assumption of a 4 per cent content, which is not official. In the prospectus of the $55,000,000 issue of debentures, dated Oct. 1, 1935, Mountain City was credited with 100,000 tons of copper metal.

[b] Anaconda Copper Mining Co. prospectus dated Oct. 1, 1935. Materially increased since the date of estimate.

[c] Not including Alaskan or Chilean properties.

[d] Including Moctezuma in Mexico, but not United Verde, since acquired, which has substantial reserves.

[e] Not including Ducktown, acquired in 1936.

[f] No data available on the reserves of Coast Copper, George Gold and Copper or Island Copper, all in British Columbia, or the long established Consolidated Copper and Sulphur Co., Ltd., in Quebec. No recent data are available regarding Newfoundland properties although substantial amounts of copper are contained at the Tilt Cove and other deposits.

[g] Plus 0 75 oz. silver and 70¢ gold (@ $35 oz.) per ton.

[h] Plus 1 41 per cent nickel.

[i] Plus 0 81 per cent nickel and $3 in gold and platinum.

[j] Plus 1 93 per cent nickel.

[k] Arbitrary assumption of 1 per cent. Company reports 2 per cent combined copper and nickel.

[l] Plus 3.86 per cent zinc, 1.28 oz. silver and $2.80 gold (@ $35 oz.) per ton.

[m] Estimated, currently treating ore higher in copper, and with nearly 3 per cent nickel.

[n] Plus 16 per cent zinc and $5 gold and silver (at prices then prevailing) per ton.

[o] Plus 5,605,515 oz. of gold.

[p] Plus 13.5 per cent zinc and 4.3 oz. silver per ton.

[q] Plus 0.87 per cent nickel, some platinum, gold and silver.

[r] Plus 61¢ gold and silver (at prices then prevailing).

[s] No change as of Dec. 31, 1935.

[t] Plus 5.8 per cent zinc, and minor amounts of lead, silver and gold.

(Footnotes continued on next page.)

Cerro de Pasco, Burma, Falconbridge or Mount Morgan, the copper market might be subject to some of the vicissitudes experienced by silver, even admitting that copper is an industrial metal with no pretensions to monetary status. Then low prices might not serve as such an efficient brake on production.

In order that a better mental picture may be obtained of world reserves, specific mention has been made of the more important mines not included in the table of foreign reserves. In the United States, however, an approximation has been made of the total for the more important properties not listed. No meticulous accuracy should be imputed to the tonnage figures of the European producers—the tonnages are the English equivalents of round figures given in metric tons. All data in the table have been secured from annual reports or other official sources, except where noted otherwise.

The tabulation shows a total of nearly 92,000,000 tons of copper metal in countries outside the U.S.S.R., while if we include the rosy estimates of probable and possible Soviet ore no less than 110,000,000 tons are contained in known reserves. These figures, with what is to

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[u] Not including 300,000 tons of 11.52 per cent zinc ore.

[v] Plus 15.8 per cent zinc, 7.7 per cent lead, 3.60 oz. silver and 0.05 oz. gold per ton.

[w] No data available on Poleo, Inguaran, Mazapil, Tecolote, Tezuitlan or Triunfo.

[x] Anaconda prospectus for $55,000,000 debentures, dated Oct. 1, 1936, give minimum reserves sufficient to produce a rate of 60,000,000 lb. for 8 years. Probable reserves are believed by others to bring the total close to three-quarters of a million tons of metal.

[y] Included in Phelps Dodge total.

[z] Plus $1.54 gold (@ $35 oz.) per ton.

[aa] No data available for Cerro de Pasco or Northern Peru.

[bb] Not exploited since date of estimate. A recent estimate is 2,013,000 tons of 3.98 per cent copper within 100 ft. of surface, and an indeterminate tonnage of about 3.15 per cent copper below.

No data available on Bolivian reserves (Corocoro).

[cc] No data available on M'Zaita (Chagres), Chanaral, Disputada, Copiapo, Gatico Naltagua, Ouancos, Poderosa or Tocopilla.

[dd] According to A. H. Rogers (January 1937).

[ee] No exploitation since date of estimate.

[ff] Plus 12.94 per cent zinc, 7.5 per cent lead, some gold and silver. No exploitation since date of estimate.

[gg] Plus some gold and silver.

[hh] Plus 0.218 oz. gold per ton.

[ii] No data available on reserves of Japanese producers—Fujita, Furukawa, Mitsubishi, Nippon or Sumitomo.

[jj] Plus 23.6 per cent lead; 14.5 per cent zinc, 18.2 oz. silver per ton, some nickel.

[kk] No data available on reserves of Bor or Majden-Pek.

[mm] Spanish reserves from "Cobre en Espana," Int. Geol. Congress, 1933. Huelva copper is credited with an additional 795,868 tons of probable ore same grade. Pyrites de Huelva is credited with an equal amount of probable ore of the same copper content. Imperial Chemical Industries, Ltd., has 4,034,455 tons of probable ore containing 40,708 tons copper metal. Rio Tinto has 166,786,000 tons positive pyrite and 77,162,000 tons of probable pyrite ore, in addition to the porphyry. St. Gobain has probable reserves of 7,700,000 tons of pyrite. San Platon has an equal amount of probable ore of same copper content. Tharsis has 106,097,338 tons of probable ore of the same copper content.

[nn] Finnish reserves from Saksela. Int. Geol. Congress, 1933.

[oo] Norwegian reserves, except Orkla, from Foslie. Int. Geol. Congress, 1933.

[pp] Mansfeld, Rammelsberg and Stadtberge reserves from Fulda. Int. Geol. Congress, 1933.

[qq] Mansfeld reserves are given as 23 sq. km., or somewhat over 20,000,000 tons of copper ore. Of this 8 km. are developed, and 15 still are undeveloped. In the 700 years of its existence 117 sq. km. have been worked out. Rammelsberg has an additional 38,581 short tons of copper metal in probable reserves.

[rr] According to C. W. Wright: Spec. Sup. No. 3, Mineral Trade Notes, U. S. Bur. Mines (Sept. 19, 1936). This company under management of Mansfeld and under heavy German subsidy, has some 480 workers, has sunk a number of shafts and drill holes and developed this estimated tonnage.

[ss] According to Poncev. Int. Geol. Congress, 1933.

[tt] Skouriatissa mine only. Mavrovonni deposit now largest producer, but no data available on tonnage.

[uu] Based on figures of Riddell and Jermain: Russian Copper. *Eng. and Min. Jnl.* (Feb. 1935).

[vv] No data available on reserves of Mindouli (French Congo); Bembe (Portuguese West Africa); Kafue (Northern Rhodesia); Umkondo (Southern Rhodesia); Tsumeb (Southwest Africa); South African Copper (Cape Province) or Northern Transvaal (Messina) Copper in Transvaal.

[ww] Including Mokambo reserves the total is more than 1,000,000 tons of metal, although Mokambo tonnage is not stated.

[zz] Plus 220,000 oz. of gold, or 70¢ per ton (gold @ $35

be reasonably expected to be developed in the near future, assure an ample supply for the rest of this century at any reasonable rate of growth in consumption.

While there has been a comparative hiatus in the development of copper ore reserves in the past few years there should soon be an addition of important tonnages in Africa, when separate companies are floated to intensively develop and exploit the various ore bodies. Canada has made remarkable gains in the past decade, and still further increases may be expected in the next few years. The only new development in the United States of which official data, though meager, are available is the Mountain City property of the Anaconda, in northern Nevada. During the past year Shattuck-Denn has developed a large tonnage of high-grade ore, unofficial estimates placing it as high as 100,000 tons of metal, together with substantial values in gold and silver.

In the older settled regions of Europe there has been much activity. Germany is bending every effort to become more self-sufficient in copper and has met with a degree of success, while Sweden and Finland have reported the development of large tonnages of ore. In Yugoslavia and Turkey exploration has been under way, but it is still too early to appraise their importance. In the tabulation of Soviet reserves, much is included that no American operator would consider as "ore" but which nevertheless may be mined if orthodox bookkeeping be completely disregarded. It seems certain, however, that Russian production will be absorbed in the home market for many years to come, and there need be little fear of Soviet bars appearing in the marts of the world in quantity.

First rank in underground reserves is held by South America, because of the Chuquicamata deposit, closely followed and perhaps soon to be overtaken by Africa. The United States comes along a very good third, followed by Russia, Canada and Europe, the reserves of Australasia and Asia (not including the U.S.S.R.) being negligible.

Estimated Cost of Producing Copper in 1935

Because of the disorganized state of the copper industry, there have been no recent extended estimates of the cost of production. A survey of the industry showed that a large number of properties were not in operation during 1935, the last year for which figures are available.

While an analysis of costs combining several years of operations is more desirable for certain purposes, it was not deemed practicable for the past few years, owing to shutdowns or widely varying rates of production. As the purpose of this cost analysis was to ascertain the relative competitive position of the various companies during 1935 in comparison with past years, no attempt was made to prorate production costs between the various metals produced, all credits being allocated to the cost of copper production. While it is realized that in actual practice

Table 3.—*Summary of World's Reported Reserves of Copper*

Short Tons

Region	Tonnage of Ore				Metal Content. Officially Reported but Ore Tonnage and Grade not Stated	Tonnage of Metallic Copper in Deposits
	Officially Reported and with Grade	Unofficially Estimated and with Grade	Officially Reported, Unofficial Estimate of Grade	Officially Reported, No Grade Stated		
United States	1,605,327,321					18,408,870
		170,000,000				2,500,000
				3,486,895		
					3,100,000	3,100,000
Total						24,008,870
Canada	102,000,695					2,101,864
			205,890,592			4,114,812
Total						6,216,676
Mexico, Cuba, Central and South America	1,364,427,707					28,989,029
		3,000,000				210,000
					240,000	240,000
Total						29,439,029
Australasia	21,746,202					446,381
Asia	4,864,983					64,384
Europe (not including U.S.S.R.)	199,754,017					2,843,360
				8,826,184		
					1,091,087	1,091,087
Total						3,934,447
Africa	632,671,102					27,235,967
					338,614	338,614
Total						27,574,581
U.S.S.R.						
Developed or drilled					6,302,347	6,302,347
Probable					4,146,449	
Possible					7,421,412	
Total					17,879,468	
World Total by classes (not including U.S.S.R.).	3,930,792,027					80,089,855
		173,000,000				2,710,000
			205,890,592			4,114,812
				12,313,079		
					4,769,701	4,769,701
Grand Total not including U.S.S.R.						91,684,368
Grand Total including U.S.S.R.						97,986,715
Grand Total including U.S.S.R. probable and possible.						109,554,576

costs for separate operations are necessarily kept against each individual metal treated, no attempt has been made at bookkeeping niceties. This is entirely justifiable from this point, especially in unit operations, where a company is mining and treating an ore body of complex metals. It would not be justified if the company were presenting a consolidated earnings statement combining widely scattered operations, each winning a different metal.

It is recognized that any cost calculations in the recent depression years may be subject to the criticism that they do not represent the true situation or obtainable costs. It is not the purpose of this analysis to represent what the individual companies could have produced copper for in 1935, having in mind the character and richness of the deposit and the adequacy and efficiency of its production facilities, but rather what they did produce copper for under handicaps. It is believed that an intelligent attempt to approximate costs is worth making, allowing the experienced engineer to interpret these figures in the light of his knowledge of the various properties and circumstances under which recent operations have been maintained.

A number of companies were excluded from consideration, and the list of these, together with the reasons, is given below.

Companies for Which No 1935 Costs Were Available

Aldermac	Experimental operation of sulphur plant only.
Bagdad	Experimental operations only.
Buchans	Controlled by Amer. Smelt. & Ref. Co.
California-Engels	Shut down.
Coast Copper	No development work during year.
Cons. Coppermines	No copper operations. Lease income from gold ores.
Kafue	No development work during year.
Mazapil	Intermittent and leasing operations.
Mount Elliott	Lease income only.
Northern Peru	Controlled by Amer. Smelt. & Ref. Co.
Opemiska	Small development work only.
Otavi	Shut down.
Poderosa	No copper operations. Small shipments gold-silver ore.
Rhodesia-Katanga	No development work during year.
Sherritt-Gordon	Shut down.
South American Copper	Shut down.
Sudbury Basin	No development work. Income from securities.
Tezuitlan	Intermittent. Leased in 1936 to Amer. Smelt. & Ref. Co.
Waite-Amulet	Shut down.
Zambesia	No development work during year.

Customs Concentrators, Smelters, Producers of Other Metals, Acids, Fertilizers, etc., for Which No Segregation of Costs Was Available

American Metal Co.	Customs, other metals and widely scattered units.
Amer. Smelt. & Ref. Co.	Customs, other metals and widely scattered units.
Black Hawk Cons.	Customs treatment and gold and silver unit.
Cons. Min. & Smelt.	Customs, other metals, acid and fertilizer.
Howe Sound	Operations in Canada and Mexico not segregated.
Mansfeld	Customs, other metals, coal, coke, acid, etc.
Rio Tinto	Also produces iron ore, pyrite, sulphur, etc.
Tennessee	Large producer of acid, fertilizer, etc.
U. S. Smelt. Ref. & Min. Co.	Customs, United States, Alaskan and Mexican operations combined.

The following companies produce other metals whose value was in excess of all operating and other charges. No attempt was made to prorate production expenses for the metals. For the purposes of this paper, and from a competitive standpoint, their copper production cost nothing. Granted a stable price for their other products, their copper would come into the market regardless of copper metal prices or agreements among the copper producers. This is also true of several in the list preceding this.

Bolidens	Produces gold, silver, arsenic, sulphur, etc.
Burma	Produces silver, lead, zinc, nickel, etc.
Falconbridge	Produces nickel and precious metals.
Hudson Bay Min. & Smelt. Co.	Produces zinc, gold, silver, etc.
International Nickel	Produces nickel and precious metals, fabricates, etc.
Mount Morgan	Primarily a gold producer.
Noranda	Produces gold, silver and rare metals, etc.

In order that the costs of a year of recovery such as 1935 may be compared with previous years, 1917, 1920 and 1929 were selected. They cover a war year, a year of postwar depression, a year of recovery and the climax of the postwar boom. They represent years of varying business activity, rather than what might theoretically be termed "normal" years, of which there have been few in the copper industry in the past generation.

The rate of operations has a decided effect upon costs, but it must not be assumed that capacity operations always bring lower costs. In widely scattered operations, low metal prices bring about the closing of a company's high-cost mines, leaving the lowest cost units in operation. Conversely, capacity operations bring back into production the high-cost units, either nullifying the reduction of costs by capacity operations at the low-cost units, or actually increasing the costs of total production. Operations at more than capacity are usually not at highest plant efficiency, while in addition labor is usually then more expensive and relatively not so productive.

The grade and character of ore treated have a decided effect on costs, and many companies have been able to keep costs down during recent times, even at a low rate of production, by mining the richer sections of their ore bodies. In most cases, however, there has been a steady diminution in the grade of ore treated over the years, and that some of them have been able to maintain such relatively stable costs of production is a tribute to their ability to consistently reduce costs per ton for all unit operations over the years.

While the subject of costs is inextricably woven with considerations of the rate of production, grade and character of the ore treated, and other

Table 4.—*Estimated Intrinsic Costs of Producing Copper**

CENTS PER POUND

(Costs for Years Prior to 1935 Adjusted to Purchasing Power of 1935 Dollar after Depreciation and All Credits, Except Where Noted)

Company	1935	1929	1924	1920	1917
Anaconda[a]	6.8	[b]	8.32	8.16	9.19
Calument & Hecla	7.96	9.65	10.19[c]	11.05[c]	8.85[c]
Copper Range	8.26[c]	12.16[c]	14.59[c]	9.23[c]	8.58[c]
Inspiration	32.66[d]	9.65	8.96	7.37	7.08
Kennecott	6.4[e]	6.54[e]	8.61	6.42	5.34
Magma	5.62	8.64	6.44	9.49	11.30
Miami	9.02	10.07	9.21	6.17	8.51
Mother Lode	4.57[f]	5.65[c]	5.83	4.49	[b]
Phelps-Dodge	7.15[e]	10.40[c]	9.99	7.94	9.02
Shattuck-Denn	11.22	10.57	8.20	12.20[c]	9.01[c]
U.V.X	7.13	7.13[c]	6.75[c]	5.11	4.17
Utah	7.50	5.58	7.24	6.82	7.49
Walker[g]	10.98	9.06	[b]	[b]	[b]
Granby	8.82	8.44	8.64[e]	8.27[c]	9.49[c]
Boleo	6.31	12.04	10.84	8.24	15.06
Greene-Cananea	4.48	5.54	10.68	8.45	11.91
Andes	7.00	7.92	[l]	[l]	[l]
Chile	6.02	6.63	8.72	6.84	12.73
Naltagua	9.19	[b]	[b]	[b]	[b]
Cerro de Pasco	[b]	5.04	6.11[c]	5.71	8.17
Bor	5.52	9.95	[b]	[b]	[b]
Indian	11.41	12.75	[l]	[l]	[l]
Mt. Lyell	7.55	8.01	7.97	9.08	12.51
Katanga	7.22	12.25	8.88	8.87	4.33[h]
Mufulira[i]	6.67	[l]	[l]	[l]	[l]
Rhokana[j]	7.13	[l]	[l]	[l]	[l]
Roan Antelope[j]	5.78	[l]	[l]	[l]	[l]
Messina	6.80[k]	9.33	10.36	13.33	[b]

[a] Estimate Butte.
[b] Not estimated.
[c] No depreciation.
[d] Charging shutdown and other expenses against two months operation.
[e] Cost per pound crediting fabricating operations.
[f] Including depletion, no depreciation.
[g] Cost of pound of copper in concentrate.
[h] Working cost only, no charges.
[i] Blister cost.
[j] Calculated on basis of electrolytic for all output.
[k] Cost credits export subsidy. Actual cost perhaps 7.7¢.
[l] Not yet in operation.

* It is realized that the fluctuations in the purchasing power of the dollar are not exactly reflected by similar changes abroad. Many of the foreign producers contract expenses in other foreign countries for supplies, equipment, some salaries, transport

technical factors, one other component in equating costs comparisons is overlooked. In any comparison involving the time element the purchasing power of money should be taken into consideration. A 15¢ cost during the World War might actually be much less than a 10¢ cost at a later date. For this reason the costs shown in Table 4 for the years prior to 1935 have been adjusted to correspond with the purchasing power of the 1935 dollar. While it is realized that the purchasing power of foreign currencies did not move in exact synchronism with that of the American dollar, the American index has been applied to them as sufficiently accurate for the purposes of this table. While some foreign indexes are available, their use would not be entirely justified, as much of the operating expenses of Latin American and African mines were incurred in other countries for the purchase of materials and supplies, services, transport, refining, marketing and administration. Adjusting the cost figures from actual to intrinsic costs reduces one of the variables in the problem, and makes an amazing change in a comparative table. In an analysis of costs based on actual currencies, it would appear that the average cost of production of the principal companies (unweighted to output) was nearly 13¢ a pound in 1917, dropping to about 10¾¢ in 1920, and remaining under 11¢ in 1924 and 1929, whereas 1935 costs were under 7½¢. Actually, when the costs are adjusted to the purchasing power of the 1935 dollar, we find the 1917 costs were about 9½¢ and 1920 costs less than 8¼¢, rising to more than 8¾¢ in 1924 and reaching 9¼¢ in 1929. Costs in the latter year were practically the same intrinsically as in 1917, a war year. During 1935 intrinsic costs were only 80 per cent of the 1917 costs. This is a remarkable showing considering low operating rates and reduced average tenor of ores treated in comparison with the earlier year. If intrinsic costs were weighted according to the importance of the producers an even greater reduction would be shown, as the big units had relatively lower costs in relation to the whole industry in 1935 than they did in 1917.

Relative Price of Copper

Just as any study of costs covering a period of years should be made on the basis of the relative or intrinsic costs, so as to eliminate one of the variables, so also should any consideration of the market price for copper metal.

In order that a clearer conception of the intrinsic value of quotations on copper metal may be obtained, rather than the actual price quotations, a chart is appended (Fig. 1) showing the price of copper in terms of its

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both ocean and rail, refining, sales and head office expenses. However for all practical purposes it is believed that the variations in the real value of money in the United States had similar counterparts abroad.

purchasing power in other commodities. The figures used are the yearly average "spot" prices for Standard copper in London, adjusted by the Sauerbeck-Statist Index. Similar adjustments have been made in the past by Warren, Black, Hay and others in studying metal and other commodity prices. Black, using the New York quotations, advanced the theory that the price for any year should be used with the index of the

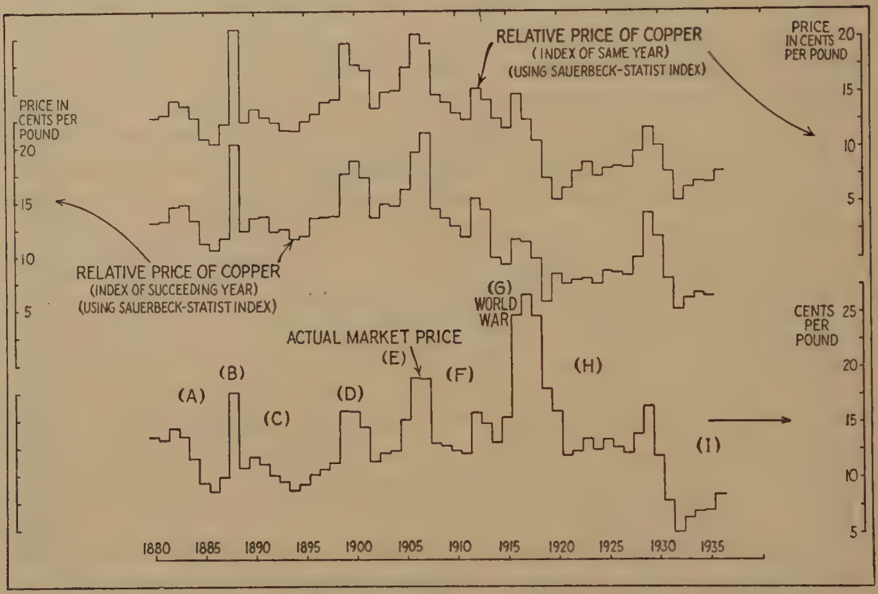

FIG. 1.—MARKET PRICE OF SPOT STANDARD COPPER IN LONDON. CONVERSIONS FROM POUNDS STERLING PER LONG TON TO CENTS PER POUND MADE ON YEARLY AVERAGE RATE OF EXCHANGE.

A. Montana discoveries and Ookiep, in Namaqualand.
B. Secretan Syndicate operations to corner world supplies.
C. Montana and Arizona.
D. Boer War.
E. Russo-Japanese War.
F. Beginning of work on western porphyries.
G. World War.
H. South America and Congo.
I. Rhodesia and depression.

following year because of forward purchasing, and the time required in manufacturing. This appears to be an unjustifiable refinement. In theory there should be a lag of a few months used, but this would vary with the character of the buying and would necessitate the use of monthly indexes. Furthermore, the prices of many of the commodities that are components of the index are subject to these same factors, so that many of the errors cancel out in part. However, almost all writers have used American prices and indexes, whereas the free world market for copper metal is in London, the American market having had numerous tariffs of varying degrees from time to time in the past 50 years. The prices

represent copper available now, not futures, and the Sauerbeck-Statist Index is available for a long period and is more representative of world influence than American indexes.

While the long-term trend of copper prices shows a distinctly downward movement in the past 130 years, since the peak during the Napoleonic Campaigns, the actual price trend during the last 50 highly industrialized years to 1929 would appear to be upward. This might lead one to the conclusion that demand was outstripping supply, that copper was becoming scarcer and the future held promise of still higher prices.

That this conclusion is erroneous may be readily seen on the chart of intrinsic prices based upon a constant purchasing power currency (Fig. 1). Actually during this period copper has steadily fallen in value; that is, in the ability of a pound or a ton of the metal to purchase a quantity of other commodities. With actual prices in 1917 far above all previous years, intrinsically copper was cheaper than in 1911; in fact, intrinsic prices for the past two decades have been on a decidedly lower plane than existed heretofore, because of the economies of modern large-scale low-cost production. The adjusted chart, whether based on the index of the same or the succeeding year, shows what the marts of the world have already sensed: the existing enormous deposits of ore and huge treatment plants presage a vast tonnage of cheaply produced metal for years to come regardless of temporary expedients of price and production control.

Summary

The recent abrupt rise in prices is not wholly justified. Part of the increased demand is due to legitimate industrial needs, but much to the accumulation of war stocks, rearmament orders, purchases in anticipation of labor troubles and speculation. The reopening of marginal producers and expansion of existing low-cost units will make available greater supplies, which will tend to act as a brake on further advances. If war clouds clear there is a possibility of much unfabricated copper finding its way back into the market to compete with new production. The rate of increase in virgin copper production will tend to taper, and secondary copper, important as it is now, will play a vastly greater role in the future.

Present world reserves of unmined copper aggregate 92,000,000 tons of metal, outside of the U.S.S.R., with indications of substantial additions during the next decade. This should be ample for the needs of the present century.

Production costs as a whole have steadily lessened over a long period of years, and intrinsic costs, adjusted to a constant purchasing currency, while not exhibiting the marked fluctuations of actual costs, nevertheless show the same trend.

While the actual price in terms of currency may exhibit a marked advance if war or severe inflation takes place, the long-term trend of intrinsic copper prices shows a steady decline in the relative value of the metal in terms of its ability to purchase other commodities. This suggests that the markets of the world are aware of the fact that regardless of temporary expedients to control production and prices, there will be many sources that will produce copper cheaply in ample volume for the world's needs.

DISCUSSION

(*Arthur Notman presiding*)

Member.—Is the consumption of copper in the United States likely to return to the 1925–1929 rate of 20 to 25 lb. per capita?

Mr. Croston replied that he thought it would but that we are now nearing the crest of virgin metal consumption.

A. Notman,* New York, N. Y., said that he felt that 15½¢ copper, barring wholesale inflation, is as dangerous as 18¢ copper was in 1929.

E. R. Lilley,† New York, N. Y., asked if increased taxes in new copper-producing countries would interfere with the authors' conclusion as to low prices.

Mr. Croston replied no, taxes should not increase the cost more than a cent. Costs, barring inflation, should be much under 10¢ even with high taxes. He believed, however, that costs would advance, a possible increase being 50 per cent of the present cost of production. A big boom might temporarily upset the market. He pointed out that a large number of properties are not now equipped for production but would be brought in with more attractive prices. In reply to another question he said that the present European demand should not be considered a normal industrial demand, but was influenced by unusual preparations for war.

A. L. Walker, Jr.,‡ New York, N. Y., asked what effect a decreased value of the dollar would have. Mr. Croston replied that if all commodities rise proportionately with inflation the price of copper will necessarily rise also, but he did not believe that the danger of inflation here was nearly so imminent as in Europe.

W. F. Boericke, Washington, D. C., referred to a paper by a Mr. Walker of London a few years ago, in which the author pointed out the immense amount of copper that might become available at a price not exceeding some exceptionally low figure such as 5¢. Mr. Croston replied that he did not think that more than a very few producers could make money at 5¢; at 7¢, yes, a few.

Someone then asked why Mr. Croston believed that so much more copper was destined to come from scrap than had so originated in the past. He pointed out that a large amount of copper was building up in industry, little of which would be destroyed but which would eventually come back on the market.

A contributor to the discussion pointed out that the African producers have been operating at well under capacity rate, and that they could reduce costs considerably

* Mining Engineer and Geologist.

† Professor of Geology, New York University.

‡ Investment Counsel.

if operating at capacity. Mr. CROSTON admitted that they could but stated that the ore they had been treating was a little above the average grade. Also later on they would have more deterioration of plant and equipment and their tax expense would also likely be greater, both of which would contribute to increase the cost.

An inquiry was then made about credits for precious metals as a factor in keeping down the cost of copper. Mr. CROSTON replied that some have enjoyed such credit but not all producers.

Someone then inquired why Mr. CROSTON believed that European war demands were of current importance in view of the fact that data so far published do not show any unusual increase in consumption in the various war-minded European countries, especially Germany. Mr. CROSTON replied that he saw no other explanation for the increased European consumption of copper. He stated that the United States consumption had lagged considerably in getting back to normal whereas European consumption in general had shown a good recovery. In the case of Germany he believed that substitutes had been used wherever possible, and that lack of foreign exchange had prevented that country from taking the copper that it otherwise would have been consuming recently. He admitted, however, that he had no data to substantiate his belief that war demands were accountable for the increased European consumption.

Mr. NOTMAN pointed out that no 1936 statistics were yet available and that general rearmament had not been actively under way until quite recently. He believed that 1937 statistics would show a big increase in copper consumption in those countries that are building up their military power.

Historical Outline of Mineral Production in Mexico

By V. R. Garfias,* Member A.I.M.E.

(Mexico City Meeting, November, 1936)

Even before the arrival of Cortes in 1519, the history of Mexico was closely linked to that of its mineral production; the mining activities of the Aztecs being thus described by Clavigero, one of the early historians:

In the mountains of Anahuac there abound in veins all kinds of metals, and infinite variety of other fossiliferous products. The Aztecs obtained gold from the countries of the Cohuijques, of the Mixtecas, of the Zapotecas and from various other places. They gathered that precious metal usually in grain, from the sands of the rivers, keeping a certain portion for the crown. They obtained silver from the mines of Taxco, of Tzumpango and others; however, this metal was not as esteemed by them as by other neighboring nations. They had two kinds of copper; one hard, which they used instead of iron to make sickles, lances and all kinds of rural and military instruments and another, soft, with which they made kettles, goblets and other receptacles. This metal abounds mainly in the province of Zacatula and in that of Cohuijques, which at present is in the kingdom of Michoacan. They obtained tin from the mines of Taxco and lead from those of Izmiquilpan, located in the country of the Otomies. From the tin they made money, and the lead, we know they sold in the market places, but we ignore the uses to which they applied it. They also had mines of iron in Tlaxcala, in Taxco and in other places; but they either had not discovered them, or did not know how to make use of the metal they contained. In Chilapa there were mines of mercury, and in other places there were mines of sulphur, alum, vitriol, cinnabar, cohre and of a white earth which they esteemed highly. As regards mercury and vitriol, we do not know of what use they were to them; the other metals they used in paintings and dyes. There were then great abundance of amber and asphalt—bitumen of Judea—on the coast of the seas and many cities of that territory paid tribute of one and the other to the kingdom of Mexico. They mounted the amber in gold, and it was only used as ornament and display by them.

With the asphalt they made certain perfumes.

The following extracts dealing with pre-Colonial gold are taken from the famous "Cartas de Relación" to Charles V, in which Cortes described the progress of the Conquest:

. . . here there came two chiefs who owned lands in the valley; one four leagues down the valley and the other two leagues up the valley and they gave me some gold necklaces of little value and seven or eight girls. . . .

. . . that day I left the City of Churultecal and traveled four leagues to some villages of the City of Cuasucingo where the natives received me very well and gave me women and pieces of gold. . . .

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* Cities Service Co., New York, N. Y.

And again:

After daybreak I went to a village, which is two leagues from here and its name is Amaqueruca of the province of Chalco and the Chief of this province gave us some forty women and 3,000 gold castellanos.

When later Cortes arrived in the Valley of Mexico he wrote the Emperor:

And upon arrival to the City of Iztapalapa, the ruler of this village came to meet me outside the boundaries and many other gentlemen were waiting for me and they presented to me from 3000 to 4000 castellanos, some women and clothing.

Regarding one of his calls on Moctezuma, he writes:

After I had left ample guards at all cross-roads I went to the palace of Moctezuma and after we had joked and talked about pleasurable things the Emperor gave me some gold jewels and one of his daughters and other daughters of the chiefs he gave to some of the gentlemen of my company. . . .

Women and gold, it will be seen, were the Aztec presents befitting these visitors whom Moctezuma thought godlike beings.

Cortes also described to his royal correspondent the wonderful work of the goldsmiths and silversmiths to be found at the market place in the Aztec capital. He mentions, "fishes with gold and silver scales, monkeys, made of gold, which can move arms, legs and tail and hold a spindle in the hand in the act of weaving." Among Moctezuma's presents to him, Cortes mentions in particular two beautifully carved "shields big as cart-wheels, one made of silver representing the moon and the other of gold, the sun." The choicest of these presents Cortes shipped to Spain as gifts to the Emperor and it is interesting to note that some of these jewels have been discovered very recently in Vienna, where they have remained since the days of Charles V among the crown jewels of the House of Austria.

While Cortes remained as an unwelcome guest in the Aztec capital and before his capture of the City, he succeeded in enlisting the help of Moctezuma in search of gold. In another of his letters, he writes:

I asked him that he should show me the mines from which gold is taken; to which the great Emperor answered that he would be pleased to do so and he then called to his presence certain of his subjects whom he sent in couples through four of his provinces where gold is produced and he asked me to give him Spaniards to accompany them so that they would see how gold was obtained and therefore to each two Mexicans I gave two Spaniards and some went to a province which is called Cozula about 80 leagues from the great City of Temixtitan and the inhabitants of this province are vassals of the said Moctezuma; and there showed them three rivers and from others they brought me samples of gold, and very good samples indeed, although gotten with little equipment because they had no other instruments but that which the Indians use. The others went to a province called Malinaltepeque, which is another 60 leagues from the Great City, which is more towards the coast of the sea. And likewise they brought me samples of gold from a big river which passes through

there. And the others went to a land to the south, and the Chief of this land is called Goatelicamat, and on account of having his land on very high mountains and rough is not a subject of said Moctezuma, and they were very well received by said chief and by the people of his land, and they showed them seven or eight rivers, from which they said they took out gold, and in their presence the Indians got some, and they brought me samples of all. The others went to a province called Tuchitebeque, which is almost on the same way towards the sea, twelve leagues from the province of Malinaltepeque, where I already said gold was found; and they showed them two other rivers, where likewise they got samples of gold.

Cortes notes that the Aztecs had not mastered the use of iron but utilized stone and copper alloys in the manufacture of implements for industrial and war purposes. And as he was at the time in immediate need of additional cannon, guided by the copper and tin fragments used as money in the province of Taxco, Cortes forthwith started to develop the mineral deposits of this region and soon his foundries were furnishing the much needed additional artillery.

Bernal Diaz, in his wonderful "Conquista de la Nueva-España," gives us this quaint sidelight on the use of copper by the Aztecs:

Besides gold the Indians of that province usually carried with them very beautiful hatchets made of copper as decorations and to be used as weapons, with painted wooden handles and we believed that the hatchets were made of low grade gold and we began to trade for them and in three days we obtained over 600 of them and we were very happy with them thinking they were of low grade gold and the Indians were even happier with the beads and trinkets we gave them, but all this was done in vain as the hatchets were made of copper and the beads we gave them were practically worthless.

Elsewhere in his book Bernal Diaz remarks that the soles of the sandals Moctezuma wore when he first met Cortes, on his arrival in Mexico City on Nov. 8, 1519, were made of gold.

While they remained in Tenochtitlan the Conqueror and his followers collected largely through the generosity of Moctezuma, valuable treasure in gold and jewels. What became of this treasure when the Spaniards were driven from the Aztec capital the night of July 10, 1520, remains to this date one of the mysteries of the Conquest. This is what Cortes writes the Spanish Emperor about this memorable flight known in history as *La Noche Triste:*

Seeing the great danger in which we were and the great damage made by the Indians each day, and fearing they would destroy that road like the others, and it being destroyed we would all die, and because by all those in my company I was asked many times to leave, and because all or the majority were injured, and to such an extent that they could not fight, I resolved to do so that night, and I took all the gold and jewels of Your Majesty that could be obtained and placed them in a room and there I delivered it in certain bundles to the officers of Your Highness, that I in your Royal name had appointed and to the Mayors and Overseers and to all the people gathered there, I begged them and asked them to help me take it out and

save it, and I gave one of my mares for the purpose, on which was loaded as much as it could carry; and I assigned certain Spaniards, my servants as well as those of others, that they follow with said gold and the rest of said officers and Mayors and Overseers and I gave and distributed the treasure among the Spaniards for them to carry. And the fort abandoned, with much wealth, belonging to your Highness as well as to the Spaniards and me, I left in the most secret way I could, taking with me a son and two daughters of said Moctezuma and Cacamacin, Chief of Aculuacan. And reaching the bridges that the Indians had destroyed, the bridge which I had made was thrown over the first canal with little labor because there were none to resist it.

And I passed immediately with five men on horseback and a hundred infantry with whom I passed swimming all the places where there were no bridges, until I reached the mainland. And leaving these people ahead, I went back, where I found that they were fighting fiercely and that there was no comparison of the injuries received by the Aztecs to that of our people, Spaniards as well as Indians of Tascaltecal who were with us, and in that way, they killed all of them and likewise many Spaniards and horses had died, *and we had lost all the gold and jewels* and clothing and many other things we had gotten and all the artillery.

The capture of Mexico City was accomplished on Aug. 13, 1521, scarcely a year after *La Noche Triste*. Soon thereafter the Spaniards started systematic mining in Taxco and Pachuca and these operations were followed by widespread explorations in search of precious metals.

The most comprehensive summary of these mining activities in Mexico and others in Hispanic America to the time of his visit to Mexico in 1802 is given by Alexander von Humboldt in his "Political Essay on the Kingdom of New Spain." In this work Humboldt quotes previous estimates of the value of the precious metals sent from Hispanic America to Europe as shown in Table 1.

He then gives his own estimate of the value of gold and silver produced in Mexico from 1521 to 1803 as follows:

	Pesos
The mines of Tasco, Zultepec, Pachuca and Tlapujahua, were almost the only ones worked immediately after destruction of Tenochtitlan in 1521, and after this memorable date to 1548. As the yearly amount of gold and silver coined at the beginning of the 18th century was over 5 millions of pesos, I estimate that from the Conquest by Hernan Cortes to 1548 the total production of Mexico at..........	40,500,000
In 1548 the production of the mine of Zacatecas began; in 1558 that of Guanajuato and almost at the same time the Patio Process was invented by Medina. From 1548 to 1600 one can estimate the production at 2,000,000 and from 1600 to 1690 at 3,000,000 making a total of..	374,000,000
In the Kingdom of New Spain there has been minted from 1690 to 1903, according to the records....................................	1,353,500,000
I estimate the contraband gold and silver not registered and taken out of the mines of New Spain from 1521 to 1803, at a seventh part of the production, or..	260,000,000

Which gives a grand total of 2,028,000,000 pesos for the value of the gold and silver produced in New Spain from the capture of Mexico City in 1521 to 1803.

TABLE 1.—*Estimates Quoted by von Humboldt of Value of Precious Metals Sent from Hispanic America to Europe*

Author	Years Covered by Estimate	Pesos
Ustariz	1492–1724	3,536,000,000
Solorzano	1492–1628	1,400,000,000
Moncada	1492–1595	2,000,000,000
Navarrete	1519–1617	1,536,000,000
Raynal	1492–1760	5,154,000,000
Robertson	1492–1775	8,600,000,000
Necker	1763–1777	304,000,000
Gerboux	1724–1800	1,600,000,000
The author of "Investigations on Commerce"	1492–1775	5,072,000,000

Elsewhere in this same study, von Humboldt presents the accompanying most interesting data on exportation of gold and silver, showing salient happenings in the mining history of Hispanic America, which is printed

EXPORTATION OF GOLD AND SILVER FROM AMERICA TO EUROPE

Years	Pesos	Important Happenings Relative to the History of the Mines
1492–1500	250,000	Discovery of the Antilles; gold placers of Cibao; expedition of Alonso Nino to the coast of Paria; voyage of Cabral. The fleets do not arrive every year to Spain; and that of Ovando was considered immensely rich, although its cargo consisted of only 2,560 silver marks.
1500–1545	3,000,000	Profit of the Mexican mines of Tasco, Zultepeque and Pachuca; Peruvian mines of Porco, Carangas, Andacava. Oruro, Carabaya and Chaquiapu (or La Paz); spoils of war made in Tenochtitlan, in Caxamarca, and in Cuzco; conquest of Choco and of Antioquia.
1545–1600	11,000,000	Discovery of Zacatecas and of Guanajuato in New Spain; hill of Potosi in the mountain range of Peru; peaceful possession of Chile and of the interior provinces of Mexico.
1600–1700	16,000,000	The mines of Potosi begin to decline, particularly from the middle of the 17th century; but those of Yauricocha are discovered. The profit from the mines of New Spain increased from 2,000,000 to 5,000,000 pesos a year; gold placers of Barbacoas and of Choco.
1700–1750	22,500,000	Profit from the placer mines of Brazil; Mexican mines of Vizcaina, Xacala, Tlapujahua, Sombrerete and Batopilas; importation of gold and silver to Spain from 1748 to 1753, 18 million pesos.
1750–1803	35,300,000	Last period of the splendor of Tasco; profit from the mine of Valenciana; discovery of the mines of Catorce and of the hill of Gualgayoc; exportation of gold and silver to Spain, towards beginning of 19th century, 43,500,000 pesos.

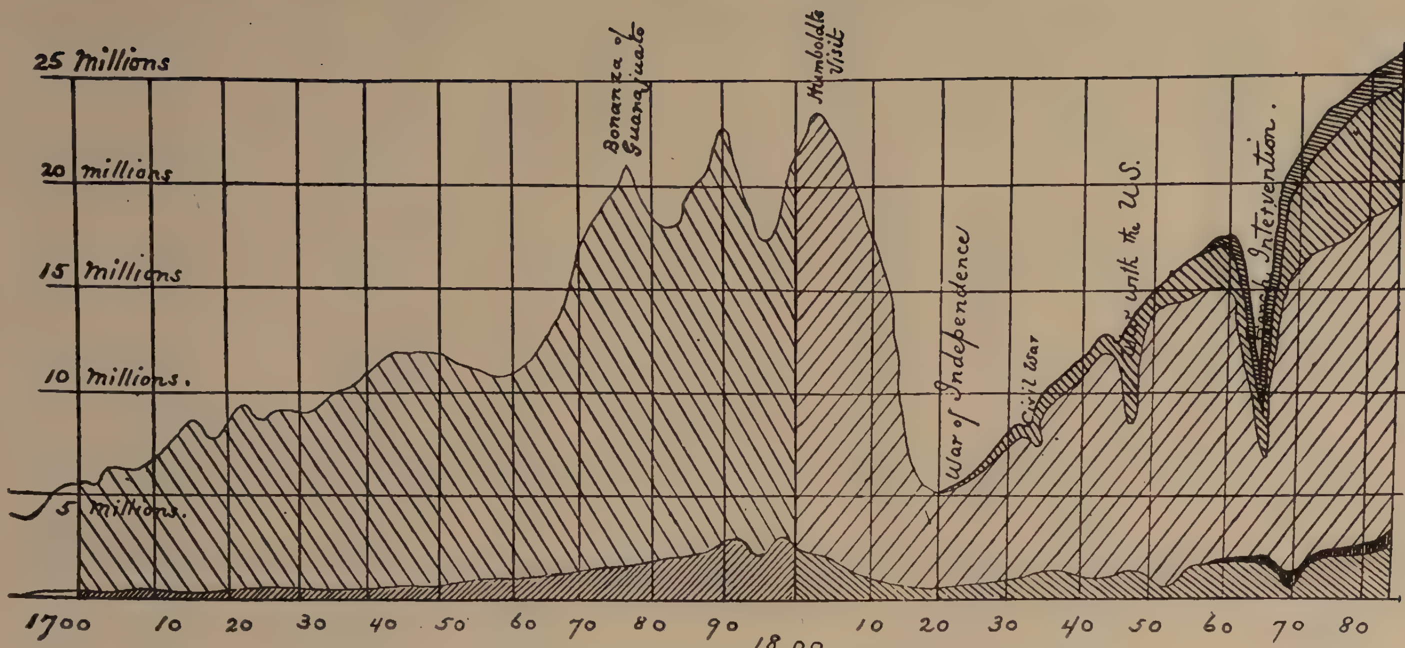

FIG. 1.—C. B. DAHLGREN'S PRODUCTION MAP, 1700 TO 1882.
Silver, in blue; Gold, in yellow.

It should be noted that the legend of the map refers to *blue* as indicating silver and *yellow*, gold. However, in all the copies of Dahlgren's book that were examined the chart is printed in black and the reader is therefore at a loss to fully understand the exact meaning of the diagram. The chart is readily understood if compared with the tables given by von Humboldt.

in its entirety although it contains some information regarding Peru and other countries.

Another interesting account of past production of gold and silver in Mexico which brings the records to the year 1882 is contained in a book published in 1883, entitled "Historical Mines of Mexico," of which C. B. Dahlgren, formerly a high officer of the U. S. Navy, is the author. Dahlgren summarizes the results of his investigations in a chart (Fig. 1).

Commenting on this map, Dahlgren says:

In general terms it can be said that from 1522 to 1879, a period of 357 years, the production of the precious metals in Mexico has been about $3,723,139,070. Of this the gold has been 0.04 to 0.08 per cent, or about $236,000,000.

This is an immense sum, but, as von Humboldt says most emphatically, "by mineralogical law much more remains." Fifty pages could be devoted to tables of the above, but the following concise statements of the Annual Coinage will be all that the public will have time to read:

Per Year	As Given by Minister of Interior 1535 to 1856 (321 Yr.)	As Given by von Humboldt et al. 1521 to 1876 (355 Yr.)	As Given by A. Garcia Cubas 1521 to 1873 (352 Yr.)
Product		$ 9,900,000	
Coinage	8,214,161	8,173,565	8,376,892
Exported		7,000,000	

Per Year	1822 to 1856	1848 to 1876	1821 to 1873
Coinage	$13,668,343	$10,247,113	$15,267,618

The accompanying map, showing the value of the productions, will illustrate, at a glance, the entire subject.

In Tables 2 and 3 the present author has brought the data regarding gold and silver to date and has included available information relating to other minerals produced in Mexico in commercial quantities. The production of cadmium and salt are not given yearly but only estimated in their aggregate.

It is fully realized that any estimates intending to show figures of commercial production and values of minerals covering over four centuries can be only roughly approximate. Nevertheless, the author believes that the estimates presented here—inaccurate as they no doubt are—if taken in connection with those made by Humboldt, Dahlgren and others will not only illustrate once more the wonderful productivity of the Mexican silver mines in the past but the great variety of the country's mineral products and the importance, in our days, of these

TABLE 2.—*Mineral Production of Mexico*[a]

1521–1936

Year	Silver, Kg.	Gold, Kg.	Lead, Met. Tons	Zinc, Met. Tons	Copper, Met. Tons	Antimony, Met. Tons	Arsenic, Met. Tons	Graphite, Met. Tons	Coal, Met. Tons	Petroleum, Bbl.	Manganese, Met. Tons	Molybdenum, Met. Tons	Mercury, Met. Tons	Iron, Met. Tons
Up to 1890	84,327,000	272,700	300,000[b]		80,000[b]				1,000,000[b]				3,000[b]	
1891	1,087,261	1,477	30,187		5,650[b]				200,000[b]				250[b]	
1892	1,250,661	1,735	47,532		7,915				350,000[b]				240[b]	
1893	1,386,479	1,862	64,000	400[b]	9,607	9[d]			260,000[b]				286	
1894	1,422,635	4,439	57,000	300[b]	11,959	80[d]			300,000[b]				300	
1895	1,456,773	8,017	68,000	500[b]	11,806	600[d]		794[c]	270,000[b]				213	
1896	1,523,803	9,583	63,000	500[b]	11,338	3,231[d]		795[c]	253,104				218	
1897	1,635,570	10,693	71,637	600[b]	11,553	5,873[d]		759[c]	359,070				294	
1898	1,743,228	12,533	71,442	1,200[b]	15,919	5,932[d]		1,365[c]	367,193				353	
1899	1,744,075	12,711	84,656	700[b]	19,427	10,382[d]		2,305[c]	409,125				324	
1900	1,766,410	12,697	63,827	1,100[b]	22,473	2,313[d]		2,561[c]	387,977				124	
1901	1,794,664	14,258	94,194	900[b]	33,943	5,103[d]		762[c]	670,000[b]	10,000			128	
1902	1,898,323	14,805	106,805	700[b]	36,357	1,218		1,434[c]	709,654	40,000			191	
1903	2,018,652	15,993	100,532	1,000[b]	46,040	2,304		1,404[c]	780,000[b]	75,000			188	9,932
1904	1,972,684	19,194	95,010	800[b]	51,759	1,694		970[c]	831,762	126,000			190[b]	23,434
1905	1,890,970	24,306	101,196	2,000[b]	65,449	1,978		970[c]	920,000	251,000			190[b]	19,674
1906	1,803,330	27,365	73,699	22,566[b]	61,615	2,418		3,915[c]	767,864	502,000			200[b]	31,062
1907	1,953,859	28,909	76,158	23,197[b]	57,473	4,615		3,202[c]	1,024,580	1,005,000			200[b]	23,082
1908	2,221,137	32,028	127,010	15,650[b]	38,173	4,046		1,076[c]	866,317	3,933,000			200[b]	23,555
1909	2,212,983	34,370	118,186	3,000[b]	57,230	3,730		1,704	1,300,000	2,714,000			200[b]	48,656
1910	2,416,669	41,420	124,292	1,833	48,160	3,730		2,571	1,304,111	3,634,000			250.6	54,698
1911	2,518,202	37,120	116,758	1,593	56,072	4,131		3,050	1,400,000	12,553,000			165.1	63,965
1912	2,526,715	32,431	105,160	1,266	57,245	1,698		3,518	982,396	16,558,000			165.2	57,832
1913	1,725,861	25,810	68,343	960	52,592	937		4,435	600,000[b]	25,696,000			165.7	12,758
1914	810,647	8,635	5,703	793	26,621	1,047		4,259	780,000[b]	26,235,000			162.4	
1915	712,599	7,358	19,971	5,806	206	739		4,189	450,000[b]	32,911,000			94.0	1,714
1916	925,993	11,748	19,971	37,449	28,411	829		470	300,000[b]	40,546,000			52.5	19,981
1917	1,306,988	23,542	64,125	45,181	50,946	2,647	1,285	420	430,820	55,293,000	73		33.1	19,119
1918	1,944,542	25,313	98,837	20,699	70,200	3,269	2,206	6,191	781,860	63,828,000	2,878	28	163.6	25,891
1919	2,049,898	23,586	71,376	11,560	52,272	471	2,246	4,023	728,374	87,073,000	2,794	2[b]	118.9	30,904

TABLE 2.—(*Continued*)

Year	Silver, Kg.	Gold, Kg.	Lead, Met. Tons	Zinc, Met. Tons	Copper, Met. Tons	Antimony, Met. Tons	Arsenic, Met. Tons	Graphite, Met. Tons	Coal, Met. Tons	Petroleum, Bbl.	Manganese, Met. Tons	Molybdenum, Met. Tons	Mercury, Met. Tons	Iron, Met. Tons
1920	2,068,938	22,864	82,518	15,651	49,192	623	2,092	3,223	715,789	163,540,000	1,137	4.5	75.7	26,034
1921	2,005,143	21,275	60,513	1,257	15,228	45	785	2,911	734,980	193,398,000	559	3.5[b]	46	34,110
1922	2,521,832	23,276	110,456	6,142	26,978	464	272	2,054	932,550	182,278,000	700[b]	3.5[b]	41.8	41,574
1923	2,824,599	24,162	155,720	18,481	53,372	490	1,403	5,489	1,261,541	149,585,000	2,246	0.5	44.7	50,694
1924	2,844,104	24,647	164,140	18,936	49,113	774	1,293	8,023	1,226,696	139,678,000	1,800	4[b]	36.7	52,448
1925	2,889,962	24,541	178,662	51,795	54,596	1,399	7,507	6,264	1,444,498	115,515,000	3,333	1.5[b]	38.7	127,492
1926	3,057,268	24,033	210,794	105,367	53,763	2,614	6,458	4,445	1,226,808	90,421,000	3,299	1.5[b]	45.4	92,982
1927	3,252,688	22,556	243,346	136,478	58,734	1,924	9,018	5,837	1,031,308	64,200,000	1,000[b]	1.5[b]	81.1	64,000
1928	3,375,966	21,645	236,486	161,747	65,505	3,578	13,000	4,972	1,022,475	50,150,000	661	1.5[b]	87.4	80,293
1929	3,381,038	20,276	248,401	174,050	86,559	2,709	9,665	5,721	1,059,956	44,688,000	650	2[b]	82.6	112,749
1930	3,278,644	21,807	242,537	124,084	73,412	3,032	9,476	5,853	1,071,658	39,530,000	732	2[b]	170.5	106,979
1931	2,676,904	19,378	210,427	120,289	54,212	2,230	7,956	3,122	922,000	33,039,000	731	3	251.4	65,156
1932	2,158,675	18,171	137,099	57,211	35,213	1,388	3,991	2,045	687,000	32,805,000	700	3	252.7	27,122
1933	2,118,182	19,836	118,460	89,339	39,825	1,559	4,697	2,685	647,000	34,001,000	573	40	154.4	77,714
1934	2,306,167	20,572	165,416	125,000	44,268	2,134	7,860	3,898	782,000	38,172,000	664	467	157.9	105,799
1935	2,351,087	21,223	184,193	136,000	39,373	3,656	9,950	7,222	1,143,000	40,235,000	3,217	687	216.4	120,000[b]
1936[b]	2,400,000	21,600	200,000	145,000	40,001	4,000	10,000	8,000	1,200,000	40,000,000	4,000	700	250	130,000
Total production.............	179,559,808	1,168,500	5,457,775	1,689,080	1,937,754	107,643	111,160	138,906	36,893,466	1,824,218,000	31,747	1,956	10,693.5	1,781,403
Rank as producer 1935........	1st	6th	3rd	5th	7th	2nd	2nd	7th		7th		2nd	4th	

[a] Data largely from U. S. Bureau of Mines.
[b] Estimated.
[c] Exports.
[d] Refers to ore; arsenic production refers to white arsenic.

TABLE 3.—*Approximate Total Production and Value of Mineral Products of Mexico*

Value expressed in terms of U. S. dollars as of 1936

Production from Beginning to Jan. 1, 1937			Price	Approximate Total Value
Silver	179,559,808 kg.	5,772,984,293 oz.	$ 0.44 oz.	$2,540,113,000
Petroleum	1,824,218,000 bbl.	1,824,218,000 bbl.	1.00 bbl.	1,824,218,000
Gold	1,168,500 kg.	37,568,163 oz.	35.00 oz.	1,314,886,000
Lead	5,457,775 met. tons	12,032,210,765 lb.	0.045 lb.	541,449,000
Copper	1,937,754 met. tons	4,271,972,468 lb.	0.095 lb.	405,837,000
Zinc	1,689,080 met. tons	3,723,745,768 lb.	0.05 lb.	186,187,000
Coal	36,893,466 met. tons	40,667,668 short tons	2.00 ton	81,335,000
Iron	1,781,403 met. tons	1,963,641 short tons	20.00 ton	39,273,000
Salt	10,500,000 met. tons	11,574,000 short tons	3.00 ton	34,722,000
Antimony	107,643 met. tons	237,309,758 lb.	0.12 lb.	28,477,000
Manganese	31,747 met. tons	69,989,436 lb.	0.40 lb.	27,996,000
Mercury	10,693.5 met. tons	23,574,890 lb.	1.15 lb.	27,111,000
Cadmium	5,000,000 kg.	11,023,000 lb.	1.00 lb.	11,023,000
Arsenic	111,160 met. tons	245,063,336 lb.	0.03 lb.	7,352,000
Tin	4,400 met. tons	9,700,240 lb.	0.50 lb.	4,850,000
Molybdenum	1,956 met. tons	4,312,198 lb.	0.70 lb.	3,019,000
Graphite	138,906 met. tons	306,232,168 lb.	0.005 lb.	1,531,000
Total				$7,079,379,000

resources. The tables show, for instance, that the total value of the mineral products aggregates over seven billion dollars and that Mexico after 400 years is yet the leading silver-producing country and that it ranks second in the production of antimony, arsenic and molybdenum and third in the production of lead. Paraphrasing Humboldt: "Mexico in 1936 remains one of the Treasure Houses of the World."

Succession of Minerals and Temperatures of Formation in Ore Deposits of Magmatic Affiliations

By Waldemar Lindgren,* Honorary Member A.I.M.E.

(New York Meeting, February, 1936)

The following pages present data accepted by many geochemists and geologists regarding the succession of minerals and the temperatures of formation in ore deposits affiliated with igneous rocks. They also present the individual views of the author on the composition of the solutions that were active and on the mode of deposition. He does not regard these views as speculations or hypotheses but as forming a well motivated and consistent theory. It is not certain, however, that all investigators will subscribe to them.

Introduction

The mining engineer may not always be interested in the field of stability and temperature of origin in the mineralogy of ore deposits, but he certainly is interested in the succession of minerals because this has a definite bearing on the understanding of the ores and on the best processes of extraction. It is proposed, therefore, to summarize in plain language the principal features of succession as shown in metallic ores. Chemical analyses of ores are always important, but the real knowledge of succession—sometimes called paragenesis—of ores has been gained only in the last 20 years, by the study of polished sections. The study of hand specimens threw much light on the subject—as shown, for instance, by Breithaupt, whose "Paragenesis," written a hundred years ago, is still useful—but the results have been modified by our slowly gained understanding of the processes of replacement.

Indications of Temperature

Melting Points.—In the first place, a distinction must be made as to whether the ore is a product of magmatic consolidation, of pegmatitization or of hydrothermal activity. Most of the ores fall in the latter class. In any case, the melting points, although of importance, are not definite indications. In a magma the oxides and sulfides are likely to be among the earliest crystalline products (magnetite, ilmenite, zircon, corundum,

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sulfides, chromite) but as they crystallize they may become more soluble in the residual magma, and some of them, at least iron oxides and sulfides, may remain in solution and separate out quite late. In acidic melts quartz is usually the latest mineral to consolidate although its melting point (1713° C.) is definitely higher than most of the silicates with which it is associated. (See table of melting points, p. 374.)

The "field of stability"—that is, the range of temperature at which a mineral can form under given conditions of components, time and pressure—is not always easy to determine. The upper limit, of course, is its melting point, above which it cannot crystallize, but this limit is ordinarily of little use because of the depression of the point by the presence of other components in the system, which almost always is a mutual solution of various substances, whether it is a "magma" or an aqueous solution. Thus, the actual melting point of quartz is very high but we know that from melts and solutions it may crystallize at far lower temperatures. The substances of low melting points give more useful data. Bismuth melts at 271° and in any ore containing crystallized bismuth it must certainly have formed below that temperature.*

Inversion Points[1,4,5,6].—Much more applicable are the "inversion points"; that is, the temperature at which a certain substance changes its atomic structure (and usually also its crystal system), without change of composition. One of the most useful inversion points is that from high-temperature or β quartz (trapezohedral-hemihedral), to α quartz, or low-temperature quartz (trapezohedral-tetrahedral), which takes place at 573° C. and like other inversion points is but little affected by change in pressure. The high-temperature quartz is stable up to 870° where it changes to tridymite. Any quartz crystal that has passed through the 573° point to α quartz presents certain characteristics that can be determined by etching. Practically all vein quartz was deposited as the alpha variety;† pegmatite may contain quartz formed somewhat below or somewhat above 573°.

Wollastonite changes at 1180° to pseudowollastonite, a form not found in nature, and the inference is justified that intrusions do not heat their limestone wall rocks (in contact-metamorphic deposits) above that point.

Another useful inversion point is shown by isometric chalcocite, which at 91° C. changes to the rhombic form. However, if it contains CuS in solid solution above 8 per cent the inversion will not take place.

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* The possibility is not excluded that the bismuth may have separated out from the solutions at a higher temperature, in liquid form.

† Lately V. B. Meen[2], by further etching tests, has found that while most of the Ontario quartz veins carry quartz formed below 573° C. some veins in the Long Lac, Sturgeon River and Red Lake districts have been formed in part above and in part below 573° C

[1] References are at the end of the paper.

Other Indications of Temperature.—The absence of dissociation in minerals containing volatiles may give useful hints. The zeolites lose water at relatively low temperature and if the occurrence indicates absence of high pressure the temperature of formation must have been low. Realgar melts at about 314° and begins to volatilize below that point. Consequently, if no strong pressure is indicated the temperature of formation must have been low.

Fluid inclusions sometimes afford quite definite information, subject to certain limitations, explained by Bowen[4]. If, as seems probable, the cavities, when formed, were filled by the solution, the relation of the cavity to the bubble may lead to important results. Thus Newhouse determined the temperature of formation of sphalerite from the Tri-State district from 90° C. to 130° C. by measuring the temperature of disappearance of the bubble[18,19].

Other authors (Koenigsberger and Holden) have attempted to measure the temperature of formation by the disappearance of color in certain minerals when heated; for instance, the purple color of fluorite or amethyst quartz. As pointed out by Bowen[4] this method is open to certain objections, so that the results may not be very reliable.

The crystal forms of minerals may often afford hints as to the temperature of deposition.

Unmixing[1,4–6,21–25].—The phenomena of unmixing have lately afforded considerable information, especially in regard to high-temperature deposits with ilmenite, magnetite, pyrrhotite and cubanite. Solid solutions of copper sulfides unmixed at 450° C. to cubanite and chalcopyrite; apparently also, in other relations, to bornite and chalcocite at 225° C. Unmixing is particularly common in the iron-titanium oxides, generally formed at high temperatures. Care must be taken, in the study of unmixing, not to confuse this process with replacement—sometimes a difficult problem.

Ore Deposits of Orthomagmatic Origin

Igneous rocks consolidate principally between 870° C. and 600° C., and, in a general way, the temperature decreases with increasing silica content[4]. The opinion has generally been held that the ore minerals (magnetite, chromite, ilmenite, sulfides) separated out early and sank in the magma towards the bottom of the magma chamber. This is no doubt true in part but later evidence makes it clear that certainly a part of the metals concerned were held by the residual magma until late in the sequence. This has been shown in connection with the New Jersey magnetites, the Adirondack ilmenites, some chromite deposits and also the Sudbury nickel-copper ores. W. H. Newhouse[15] has found that chalcopyrite in igneous rocks always separates out as one of the latest products.

The sulfide ore minerals are few and simple: pyrite, pyrrhotite, pentlandite, sphalerite, chalcopyrite. The Sudbury case illustrates how difficult the solution of the problem may be. Theories have been suggested involving an early magmatic separation and consequent sinking, followed by remelting and injection (Kiruna). Of course, this involves additional heat and also reduction of the high melting points by admixed solutions. At Kiruna, fluorine, in apatite, probably causes the lowering of the melting point.

Pegmatites

In the pegmatites, which probably began to crystallize about 600° (Bowen[4]), the mica, orthoclase and quartz were almost certainly the earliest products. Later came magnetite, often well crystallized but rarely in economic quantities; this was followed by tourmaline, cassiterite and a host of minerals of the rare earths, the sequence probably ending with scant sulfides and arsenides, such as löllingite, arsenopyrite, molybdenite, very rarely chalcopyrite, sphalerite and galena. In many cases, the temperature sank to more moderate hydrothermal conditions during which zeolites and other low-temperature minerals were formed.

Character of Solutions during Transition from Igneous to Hydrothermal Conditions

We have gradually obtained some information as to development of the hydrothermal fluids, largely by the excellent work of such men as Bowen, Fenner, Morey, and others connected with the Carnegie Geophysical Laboratory.

Bowen has lately presented a very good picture of the constitution of the residual magma[36]. It is well known that the volatile products given off during the consolidation of surface lavas are of acidic reaction and besides water contain sulfur, fluorine, chlorine (as HCl) and other elements; with these are entrained more or less of practically all metals and even some silica. Ascending as fumaroles they may deposit magnetite and sulfides, near the surface, but they do not ordinarily form ore deposits.

In intrusive bodies the residual magma is an aqueo-igneous melt of SiO_2, K_2O, Al_2O_3, Na_2O, which is in condition to produce pegmatite. Finally, there takes place by fractional crystallization a separation between the silicates and the volatiles. The pressure increases and the volatile extract, still of acid reaction, may escape and is now in condition to attack any minerals it may encounter. "It is probably these acid solutions that are the principal agents of contact metamorphism, of vein formation, of replacement and metasomatism in general."*

* Bowen[36], p. 123. The writer cannot agree with Dr. Bowen when he says (p. 127) that "most ore deposits are formed in depth through the agency of acid solutions."

The transport of sulfides and oxides of the heavy metals in acid solutions, Bowen remarks, is no problem at all, but the transport of most of them in alkaline solutions is indeed a very serious problem. The transportation of silica also offers difficulties for the chemists; recent work seems to indicate the possibility of volatile transfer in some form of silicic acid; the small quantity of chlorine and fluorine is quite inadequate for the transport of the large amounts of quartz that occur in even the most deep-seated ore deposits. And, it is a fact not appreciated by all that in the pyrometasomatic (contact-metamorphic) deposits that are closest to the acid emanation a great deal of silica has been carried in from the intrusive.

The present writer believes that from the point of consolidation (the melting point being tremendously depressed by the presence of water) the silica is taken up by the water in colloidal solution[34].

According to V. Lenher and M. T. Schrenk[34] transparent quartz crystals are slowly gelatinized at 500° C., and it may be recalled that Koenigsberger obtained chalcedony from solutions about 300° C. Aside from haloid compounds (and possibly water gas at high temperature) this seems to be the only way in which silica can be transported in aqueous solution. It is held by many, including Bowen, that the critical phenomena at 364° C. are of little importance in ore deposition.

The acid solutions, with their load of metals and silica, now ascend towards the surface. True, some physical chemists believe that, once condensed, they have lost the power to ascend; but it seems to be a stubborn fact that they *do* ascend. On their way they attack various minerals of the igneous rocks or their surrounding geological bodies, rapidly become neutral or alkaline and remain thus to the surface.

The question now becomes urgent—how to keep the heavy metals in solution when the change to alkalinity occurs.

It is, of course, quite impossible to demonstrate "ex cathedra" what happens in such a complicated system. It has to be worked out by long and painstaking experimental work. What follows, therefore, is simply the views of the writer as to the most probable processes.

It is premised that a given deposit has generally been formed under conditions of gradually decreasing temperature and pressure. Local reversals, temporary increase of temperature, new sources involving change in composition of the waters, may, of course, have happened, but only the normal course will be considered here. As these views necessarily involve colloidal processes, it is very likely that strong objections will be raised. The average geologist has been raised on the theory of electrolytic solutions, and when colloidal phenomena are mentioned they will usually be met by one or more of these statements: (1) Colloids are unstable at high temperatures; (2) colloids are coagulated by ever-present electrolytes; (3) well developed crystals cannot form from

colloidal solutions. Of course, all these statements are true only to a very limited extent[35,36].

The acid extracts separate, according to Bowen, from the residual magma by a process of boiling. They probably contain sulfur, sulfuric acid, possibly SO_2, certainly H_2S, also much H_2O. H_2SO_4 may in some degree be fixed by alkalis and the alkaline earths and in such form may be stable during the ascent of the solutions; but the sulfates, as well as the chlorides, of the heavy metals, can probably not persist in the presence of H_2S, but tend to be precipitated as sulfides. As the alkalis are present in abundance, particularly Na, it seems probable that alkaline sulfides will form and that these will combine with the heavy metals to double sulfides.

The precipitation of sulfides may be inhibited by the presence of CO_2 together with the H_2S; it is also inhibited by any acid formed during the decomposition of metallic chlorides, say lead chloride. Such facts would explain why all the heavy metals are not precipitated by H_2S at once.

Fenner says[35] that "the presence of alkaline sulfides in such a solution is regarded as a probably essential factor in increasing the solubility of the heavy metal sulfide to a sufficient degree to effect . . . transportation and deposition by the long continuance of migration along appropriate channels."

The precipitation in this complex system of double sulfides will take place, in part dependent upon the solubility of the sulfides[7], in part upon the changes in temperature and pressure, disturbing the equilibrium of the volatiles.*

The exact order must be determined by future chemical work. In the main, as indicated by Wells[3] and by Newhouse[14], it is a factor of solubility and the order on the whole seems to follow the Schurmann series (Hg, Ag, Cu, Pb, Zn, Ni, Co, Fe). At any rate, it is certain that the sulfide deposition in the great majority of cases begins with pyrite, the most soluble, and closes with cinnabar, the most insoluble.†

Thus, during their ascent, the hydrothermal solutions will gradually deposit their load in a well defined order and when they emerge at the surface they are quite dilute and contain alkaline sulfates, carbonates, chlorides and borates, besides silica, H_2S, CO_2 and a residuum of alkaline sulfides of mercury, arsenic and antimony.

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* "A mixture of two metallic salts yields, by fractional precipitation, an initial precipitate containing the sulfides of both metals, but, as a rule, if the mixture is heated or allowed to stand, one sulfide largely or wholly dissolves[3]."

† Weigel's data on the solubility of sulfides refer to water. Of course, the solvent in the hydrothermal fluids is not pure water. Other factors such as the electrode potentials and the hardness (harder minerals early, soft minerals late) enter into the case, but the exact importance of these is as yet not determined. See B. S. Butler and W. S. Burbank[16] and G. Gilbert[11]. The mingling with surface waters is another factor.

Further discussion about the various minerals deposited during the paragenesis will be found on later pages.

The writer believes then that the alkaline solutions obtained by the neutralization of the acid emanations from the magma contain alkaline chlorides and sulfates, sulfates of alkaline earths, bicarbonates of the alkaline earths and of iron; further, silica sols, and sols of stannic and tungstic acids, free carbon dioxide and hydrogen sulfide. And further, he believes that sodium sulfide and possibly polysulfides result from the action of H_2S on sodium carbonate. Sols of ferric hydroxide and ferric oxide may be present, but under the influence of H_2S and Na_2S, the oxy-salts of the heavy metals are transferred into colloidal sulfide solutions, the silica sol acting as protecting agent. This is the only way in which the sulfides of the heavy metals can be transported in such solutions.

Changes in the equilibrium of this solution (pressure, temperature, concentrations, dilution and other factors) cause the sulfide sols to coagulate, some singly, others in combination, others forming solid solutions (which afterwards may unmix). Ordinarily this takes place in a definite order expressed by the paragenesis, and determined by solubility, electrode potential and other factors. Experimental work on less complex systems can no doubt furnish a complete solution of this problem, in spite of the fact that colloidal solutions are notoriously capricious in their action. The precipitated gels will either form fine-grained metacolloids or well developed crystals may form directly from the sols, the latter mainly at higher temperatures. Very slight changes determine one or the other mode, as well illustrated by the layers of coarse quartz crystals alternating with microcrystalline chalcedonic layers in geodes of volcanic flows.

If the temperature is high enough the deposition generally begins with silicates such as chlorite, sericite, tourmaline, etc., by the action of the silica sols and boric acid on salts of the alkaline earths and iron. Later, cassiterite and wolframite form, followed by the sulfides, and the series ends with tellurides and gold.

Colloidal chemistry records the ease with which sulfide sols can be produced, especially by the reduction method and particularly by H_2S. Long ago (1888) Winssinger[32] prepared many sulfide sols by the action of hydrogen sulfide gas on very dilute solutions of salts of Pt, Pd, Au, Ag, Pb, Bi, Fe, Ni, Co; also, by slight modification from salts of Mo and W. Zinc sulfide sol was prepared by passing hydrogen sulfide into a suspension of zinc oxide.

It is hardly necessary to emphasize the recent work of Tolman and Clark, Clark and Menaul[30] and Young and Moore, which has so convincingly demonstrated the same process. Nor is it necessary to insist on the incomparably greater solubility as sols than in water in the case of many substances. Boydell[31] states that an aqueous solution of ZnS

contains only 3.3×10^{-18} grams per liter while the sulfide precipitate, peptized by H_2S, contains 0.159 grams per liter.

Boydell[33] showed that the deposition of a metalliferous vein would be a very protracted operation if caused by the precipitation from hypogene true solution; but much less so from colloidal solutions. The limitations which he pointed out, if we assume precipitation from true solutions, are: (1) extreme dilution, (2) necessity for maintaining a slight supersaturation, (3) continuous supply of precipitants, and (4) the frequent alteration of constituents and precipitants.

It is not meant to imply that all minerals were deposited as colloidal precipitates, for the reactions are very complex; much depends on the power of crystallization. Lead sulfide, for instance, may be precipitated by H_2S directly as minute cubes. But, generally speaking, the writer believes that most sulfides passed through a colloidal state, although it may have been short.

Paragenesis in Hydrothermal Ore Deposits

The writer believes that all hydrothermal veins and replacement deposits were formed at temperatures between 600° C. and some point well below 100° C.*

In a paper entitled "Magma, Dikes and Veins," a general paragenesis for hydrothermal (and contact-metamorphic) deposits was attempted[8], and is here introduced again with some corrections and additions. It begins with the oldest minerals:

1. Quartz (continued), chlorite, tourmaline, lime-iron silicates, sericite, albite adularia, barite, fluorite, siderite, rhodochrosite, ankerite, calcite (continued).
2. Magnetite, specularite (sometimes a little later); uraninite.
3. Pyrite, arsenopyrite, cobalt and nickel arsenides.
4. Cassiterite (sometimes preceding pyrite), wolframite (scheelite), molybdenite (?).
5. Pyrrhotite, pentlandite, chalcopyrite, stannite, bismuthinite (?).
6. Sphalerite, enargite, tennantite, tetrahedrite, chalcopyrite, bornite, galena, chalcocite, stromeyerite, argentite, ruby silver, polybasite, chalcopyrite, lead-silver sulphantimonites, silver, bismuth, electrum, tellurides, native gold.
7. Stibnite, cinnabar.

The latest deposition often closes with a little calcite and quartz. There is, of course, more or less overlapping and sometimes reversals (e.g., galena followed by sphalerite). If we begin to test this general list against similar lists for various deposits, the interesting fact soon becomes apparent that practically the same paragenesis holds for contact-metamorphic, hypothermal, mesothermal and epithermal deposits. Some

* In contact-metamorphic deposits, close to the igneous contact, the temperature may have exceeded 600° C.

minerals or classes of minerals may be absent but, as a rule, the relative position is the same. We may select some typical cases:

Caracoles, (Bolivia). Hypothermal. Chlorite, sericite, tourmaline, apatite, quartz, cassiterite, wolframite, pyrite, sphalerite, bismuthinite (?). Repeated quartz, ankerite, pyrite.

Potosi (Bolivia). Mesothermal. Quartz, pyrite, arsenopyrite, cassiterite, sphalerite, chalcopyrite, stannite, tetrahedrite, andorite, ruby silver, jamesonite.

Chocaya (Bolivia). Epithermal. Quartz, pyrite, cassiterite, chalcopyrite, stannite, sphalerite, galena, tetrahedrite, jamesonite.

Smuggler (Colorado). Epithermal. Quartz, pyrite, arsenopyrite, rhodochrosite (?), chalcopyrite, sphalerite, chalcopyrite, galena, tetrahedrite, polybasite, chalcopyrite, gold, calcite.

Butte (Montana). Mesothermal. Copper veins: quartz, pyrite, sphalerite, enargite, tennantite, bornite, chalcopyrite, chalcocite. Silver veins: manganese carbonate, calcite, fluorite, pyrite, chalcopyrite, sphalerite, galena, argentite.

Cobalt (Ontario). Mesothermal. Calcite, diarsenides, sulfarsenides of nickel and cobalt, arsenide and sulfarsenide of iron, monoarsenides, silver, bismuth. Later scant sulfides of Fe, Cu, Pb.

Coeur d'Alene (Idaho). Mesothermal. Siderite, sphalerite, galena (quartz).

Colquijirca (Peru). Epithermal. Chalcedony, kaolin, dolomite, ankerite, barite.

Pyrite, sphalerite, enargite, tennantite, bismuth minerals, galena, chalcopyrite, stromeyerite.

Last phase: chalcedony, specularite, marcasite, chalcopyrite, galena.

Goldfield (Nevada). Epithermal. Silica, kaolin, alunite, pyrite and marcasite, famatinite, tennantite, sphalerite and wurtzite. Bismuthinite, goldfieldite, Au, Ag tellurides, gold[9].

It is thus quite evident that the differences of temperature within the hydrothermal range, say from 600° C. to 50° C., do not make much difference in the relative paragenesis.

Naturally, there are differences in the mode of deposition. In the epithermal deposits the sulfides are crowded within a shorter vertical distance; they are "telescoped," using Spurr's expression; the base-metal sulfides do not predominate as they do in mesothermal and hypothermal deposits, and gold, tellurides, antimony and mercury are more prominent. In the mesothermal and hypothermal classes the sulfides are spread over a larger interval and zoning is more apparent. In epithermal deposits and, the writer believes, only in such deposits, a recurrence of H_2SO_4 acidity is sometimes noted. It is not intense but marked enough to produce alunite and dickite (a kaolin mineral).*

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* Lately, L. C. Graton and S. I. Bowditch, in an important paper presented at the New York (1935) meeting of the Society of Economic Geologists, described the effects of hypogene acid solutions in the Cerro de Pasco deposit, S. A. There is here a widespread older sericitization, indicating originally alkaline solutions upon which, by a later change to acidity, the alunitization is superimposed. These interesting observations, however, may also be interpreted to conform with the idea of originally acid emanations soon neutralized to produce widespread sericitization, and again, nearer to the surface, becoming acid enough to produce alunite.

Deposition of Iron

In any case, pyrite is the earliest of the sulfides. Now, iron may be carried as a colloidal sulfide or as double sulfide of Na and Fe. It may also, if there is enough CO_2, be carried as a bicarbonate; not unlikely, also as a ferric oxide sol, which may be quite stable; part of the iron may come down earlier as an oxide or as a silicate. But pyrite predominates as a rule and, as stated already, it is by far most abundant in the deposits indicating high temperature.

Pyrite is very easy to reproduce[20]. It may be obtained by treatment of siderite, magnetite or specularite with H_2S between 100° C. and red heat; by treating ferric chloride with H_2S in a red hot glass tube; by crystallization of colloidal FeS, in the presence of H_2S under pressure. Allen, Crenshaw and Johnson obtained it from ferrous sulfate by prolonged heating in a closed glass tube, with sulfuric acid and hydrogen sulfide.

The many ways in which the mineral may be formed in nature at low temperatures are well known. Alkaline solutions are most favorable for the synthesis of pyrite while marcasite forms best in acid solutions.

Ordinarily, in hydrothermal deposits, pyrite forms at the highest initial temperature. With sinking temperature, zinc, copper, and other sulfides are precipitated; towards the end the double sulfides, such as sulfarsenites and sulfantimonites of copper, lead and silver come in.

Wherever the arsenides and sulfarsenites of iron, cobalt and nickel are present, they are among the earliest minerals, usually following closely after pyrite. The arsenides of copper in the Lake Superior copper mines are a little difficult to place, but from the data given in *Professional Paper* 144, U.S. Geological Survey, it would appear that the arsenides poor in copper followed the nickel and cobalt arsenides and were again followed by native copper. At any rate, the native metals certainly come late in the general paragenesis; they are copper, silver, bismuth and gold. The same applies to the tellurides: Altaite (Pb), tetradymite (Bi), coloradoite (Hg), hessite (Ag), petzite (Au, Ag), sylvanite (Au, Ag) and calaverite (Au). They are practically always closely associated and in part contemporaneous with native gold.

Deposition of Tin

It is not known in what form tin is present in the magmatic emanations. It has been supposed to occur as a fluoride or a chloride and it has also been assumed that, by reaction of the fluoride with H_2O, SnO_2 and HF are formed. When we consider that cassiterite is commonly present in colloform aggregates in epithermal and mesothermal deposits, it seems more likely that it enters early into the neutralized emanations as a sol, either of SnO_2 or of α or β stannic acid, and that the SnO_2 is

deposited by coagulation of such sols. As well known, the SnO_2 acts very similarly to SiO_2. A small part of the tin occurs as sulfide, in stannite. Some of the tin therefore may be precipitated as a sulfide or transformed into a sulfide sol.

Deposition of Tellurides

The tellurides occur abundantly in some epithermal veins but are by no means unknown in mesothermal veins. And, again, they are widespread in certain hypothermal gold veins like those of Ontario and Kalgoorlie. Regarding the occurrence of tellurides in the Ontario (and Quebec) veins we have much valuable information in many papers by Ellis Thomson (published in University of Toronto Geological Studies from 1922 to 1935). It is perfectly clear that they are very late, although in some places gold seems to be still later. They occur in veinlets cutting quartz or ankerite, and in places are associated with a little chalcopyrite and galena. It would seem to be a fair conclusion that they were deposited at the lower temperature limit for the deposit in which they occur.

Ellis Thomson[10] gives the following succession for the tellurides at the Robb-Montbray property, Quebec: Pyrite, chalcopyrite, pyrrhotite, sphalerite, krennerite, tetradymite, altaite, petzite, coloradorite, native gold.

Although tellurides rarely occur together with selenides, it is probable that the latter also appear at a relatively low temperature and late in the sequence. Gold selenide probably occurs in nature but is not known as a definite mineral.

From the occurrence of tellurium and selenium in volcanic sulfur it is concluded that hydrogen telluride and hydrogen selenide must form part of the magmatic emanations, together with the predominating hydrogen sulfide. At any rate H_2Te and H_2Se precipitate many tellurides and selenides of the heavy metals and no doubt are as potent in dispersing them to form sols as is H_2S. These sols are stabilized by silica and by Na_2S and, finally, become coagulated and crystallize towards the end of the deposition. Both gold and silver are soluble in tellurium (melting point 452° C.) and their eutectic points lie respectively at 415° C. and 351°, but these eutectics do not occur in nature. Tellurium and tellurides, also H_2Te and Na_2Te, precipitate gold from its solutions. Few of the synthetic experiments on tellurides are applicable to natural occurrences, most results being obtained by fusions and sublimation. Neither hessite nor calaverite nor sylvanite has been artificially made.* It is

* Experiments by Pellini and Quereigh, quoted by Doelter[20], seem to show that a compound of the composition of calaverite can be obtained by fusion of Au and Te. Also that hessite and petzite were obtained by Margottet by passing telluride vapors over metallic silver or a gold-silver alloy in a glass tube at red heat. Altaite (PbTe)

apparent that we have no very definite information as to the mode of deposition of the gold tellurides. The following condensed abstract presents the opinion of Lindgren and Ransome[26] in their report on the Cripple Creek district:

Basing his opinions on the work by Lenher and Hall, Van Hise held that the probabilities were decidedly against the view that gold, silver and tellurium were transported as tellurides. These authors (L. and H.) found no solvent whatever for the tellurides which would not break up these compounds with the production of salts of tellurium and native gold. They found that practically all metallic tellurides, native tellurium and hydrogen telluride rapidly reduce gold from its solutions, forming metallic gold. The conclusion is that tellurium and gold traveled in separate channels. A possibility was pointed out that $AuCl_3$ and $TeCl_4$ might be able to coexist and that when these solutions entered trunk channels, the sulfides there contained might reduce both gold and tellurium together and thus produce the tellurides of gold. Lindgren and Ransome do not agree with this, holding that gold and tellurium existed together in the vein solutions. Experiments by W. F. Hillebrand showed that, while a solution of bicarbonate of sodium was without effect on calaverite even when heated for many hours at 150° C., the same solution, more or less charged with hydrogen sulfide, would attack calaverite even with an exposure of one or two hours at room temperature and the amount of Te and Au dissolved corresponded fairly well with the composition of calaverite. We (L. and R.) incline to the belief that the gold tellurides were deposited as such and that they were precipitated by supersaturation, due to various physical changes. We believe that the gold and the tellurium are emanations from a cooling magma. We believe that the temperatures of deposition varied from 200° C. to 100° C. We have reached a point where further progress depends on experimental work with solutions at high temperature. The gold tellurides are practically confined to the open fissures and the conclusion was reached that the walls were not easily permeable to the gold-telluride solutions. The silica was in all probability contained in the waters in colloidal, easily soluble form and it is at least a legitimate subject for inquiry whether or not the sulfides and the tellurides were dissolved in the same manner. It is well known that the various sulfides of the metals, as well as some of the metals themselves, particularly gold, can exist as colloidal suspensions. Under certain conditions, hydrogen sulfide may produce colloidal suspensions of sulfides instead of precipitates and these, again, may be coagulated by a sufficient amount of an electrolyte. On the other hand

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was obtained by fusing Pb and Te at 500° and sublimation in nitrogen current. Coloradoite (HgTe) was obtained by Margottet by sublimation method, uniting vapors of Hg and Te at 800°. All these experiments require further investigation.

the presence of a fairly small quantity of a gelatinous colloid may prevent this coagulation. How far crystals can be produced from such a colloidal solution is not certain but that quartz can and does crystallize from them seems fairly well established by observation at siliceous springs.

The opinion of Lindgren and Ransome in the above abstract is clearly that the gold telluride existed in the solutions containing sodium carbonate and sodium sulfide and that they existed as colloidal double salts of the sodium sulphide and gold telluride. Please note that the report referred to was published in 1906. It is a severe indictment of the science of ore deposition that practically no progress has been made in this subject during 30 years.

At any rate, the writer desires to point out that some of the conclusions in this present paper are not new but were really reached a long time ago. Nevertheless, it must be acknowledged that the mode of deposition of gold tellurides and gold-silver tellurides is not as yet fully elucidated.

H. Borchert[27] has recently published some interesting statements on inversion points and temperatures of formation of tellurides, showing that calaverite has an inversion point at 184° expressed by a lamellar structure apparent on etching. Heating to 184° and beyond produce notable changes in this structure. He shows that the Cripple Creek calaverite is of the high-temperature form, which crystallized above 184°, while the calaverite from Kalgoorlie and Sacaramb belongs to the low-temperature type formed below that temperature. Hessite, the silver telluride, has frequently a lamellar structure, which upon heating to 149.5° becomes irregular. He holds that the temperatures of formation of hessite lie between 150° and 184° C. If these conclusions are confirmed, we would have an important geologic thermometric point and it would be concluded that the closing temperature at Kalgoorlie (a hypothermal deposit), for instance, was about the same or lower than that of the main telluride deposition at Cripple Creek (an epithermal deposit).

Deposition of Silver as Sulfide and Telluride

These relations of the tellurides are evidently similar to those of argentite and acanthite (both Ag_2S), for it is known that argentite has an isometric structure only above 180°. Isometric argentite is thus the high-temperature form. Twinned structure and anisotropism develop below 180°, and rhombic acanthite is thus the low-temperature form, below 180° C. This seems to account for the prevalence of argentite and acanthite in epithermal deposits.

Silver sulfide is one of the latest minerals to form, generally preceding gold and often appearing intergrown with native gold. It is a common mineral in epithermal veins, much less common in mesothermal veins, and

probably does not occur at all in the hypothermal deposits. Argentite is easily synthesized by the action of alkaline sulfides or hydrogen sulfide on silver salts. Not all of the silver is retained in solution to the end. More or less silver is precipitated with the galena, with tetrahedrite, tennantite, etc., and enters into these minerals as part of a solid solution. In galena, it is almost wholly ejected by unmixing, separating as small dots of argentite in the lead sulfide. Probably the other sulfides also carry down with them certain small amounts of silver. Later, much of the silver may be precipitated as sulfosalts of arsenic, antimony and lead. Finally, come argentite and acanthite, referred to above.

Argentite and most of the other silver minerals are insoluble in alkaline carbonate[28]. Calcite and rhodochrosite do not precipitate silver from dilute solutions of silver carbonate. The sulfosalts of silver are unstable in hot alkaline sulfide, and leave Ag and S in the residue.

According to Freeman[29], silver as well as zinc and lead forms double sulfides with Na_2S, which are slowly broken down by water and yield colloidal suspensions of Ag sulfide and solution of Na_2S.

Tellurium precipitates hessite (Ag_2Te) from silver salts. Selenium, at higher temperatures, reduces gold chloride to metallic gold.

Clark and Menaul[30] did not succeed in dispersing argentite by passing H_2S for two months through a water bottle containing the mineral. But stable silver sulfide sols have been made by passing H_2S through solutions of silver salts, the colloid being stabilized by gum arabic; probably silica sol would have been equally efficient. One of the sols contained 0.6 grams Ag per liter.

The writer holds that silver sulfide has probably been deposited by sols stabilized by Na_2S or by SiO_2.

Deposition of Gold

The case of gold is of particular interest because so much of it is deposited in metallic form. Native gold, as a primary constituent of igneous rocks, is rare but has been described by Moericke in pitchstone from Chile and by G. P. Merrill in a fresh granite from Sonora, Mexico. It also occurs occasionally in quartz of pegmatites. It occurs sparingly in contact-metamorphic deposits but quite abundantly in hypothermal and mesothermal deposits; also in epithermal deposits but not always in visible form.

Compounds of gold are few: there are the tellurides already referred to; probably a selenide, which, however, has not yet been definitely identified; also some compounds with mercury, bismuth and palladium that perhaps rather should be referred to as alloys. The native hypogene gold always contains more or less silver alloy reaching 50 per cent in electrum. About the most stable gold salt is the auric chloride ($AuCl_3$),

which forms the starting point of most speculations as to the state of gold in natural solutions. It is said to be stable up to 450° in the presence of NaCl. Other authors give 254° and 370° C. as the temperature at which dissociation begins[38]. However, gold salts are easily reduced even by the mildest reduction agencies; metallic gold or colloidal solutions of gold result. Gold oxide is fairly stable at lower temperatures, but, like the sulfides, does not occur in nature. Au_2S_2 decomposes at about 200° C. Cold solutions of alkali-hydrosulfide readily dissolve Au_2S_2. Alkali polysulfides also dissolve Au_2S_2 rapidly. This sulfide is completely decomposed at 240° C. Sodic aurosulfide (AuNaS, $4H_2O$) results when metallic gold is heated with Na_2S and sulfur. It is a crystalline salt. AuS is formed (with HCl and H_2SO_4) when H_2S is passed through cold $AuCl_3$ solution. It is a black powder, which dissolves in Na_2S solution and is thereby reduced to Au_2S. It also readily forms colloidal solutions in water. When left in contact with $AuCl_3$ it turns into metallic gold. Auric sulfide (Au_2S_3) decomposes into metallic gold and S at 200° C. It dissolves in sodic sulfide, forming sodic auric sulfide. Hydrogen sulfide precipitates Au_2S, but mixed with gold and sulfur, from a hot $AuCl_3$ solution. Other authors say that brown metallic gold is precipitated by H_2S from hot solutions of $AuCl_3$. Altogether, it seems very doubtful whether either $AuCl_3$ or a gold sulfide can exist in hot hydrothermal solutions. On the other hand, it is probable, although definite information is lacking, that gold sols are stable at quite elevated temperatures when a suitable protecting agent is present.

Hatscheck and Simon, for instance, have experimented with gold sols protected by silica sol and found them very stable. But, the reduction agents that have been used for the preparation of gold sols (oxalic acid, carbon monoxide, phosphorus and various organic compounds) are not likely to occur in natural solutions.

That gold is soluble in Na_2S and NaHS is asserted on the basis of well-known experiments by G. F. Becker and V. Lenher. S. P. Ogryzlo[39] states in a recent paper that it is not soluble (except in traces at high temperatures) in Na_2S, but that it is soluble in NaHS at high T and P; also, to some extent at room temperature. However, the earlier data were quite definite.

Freeman[29] fused with Na_2S a nugget of gold with some quartz attached. The product dissolved in water, giving a yellow solution. Upon standing exposed to the air for 15 days, gelatinous silica was deposited and, on its upper surface, brown gold was precipitated.

H. L. Sulman[40] says that metallic gold is readily soluble in solutions of alkaline sulfides and especially in polysulfides. The gold is present, he holds, either as a double sulfide of gold and alkalis, or as sulfide, or as a gold sol. The same author also describes a massive pyritic gold ore from Hungary, which assayed $800 per ton but gave no gold on crushing

or panning. When the pyrite was dissolved by HNO_3 a delicate sponge of gold was left, which he justly concluded seemed to prove that the pyrite and the gold was precipitated simultaneously. There are many such ores among the epithermal deposits: El Oro, Mexico and Waihi, N.Z., are other examples. R. E. Head[37] has recently investigated the occurrence of gold in pyrite and other sulfides as well as in tailings and concentration byproducts. He does not deny the possibility that gold may occur in solid solution in pyrite, but shows that most of it is present as fine metallic particles and thinks that "coating" of the gold by iron hydroxide and other substances accounts for most of the failures of the metal to amalgamate and float. Recently, G. Bürg[41] discussed this question in detail, in part basing his conclusions on certain Colombian pyritic ores. He finds that the gold is forced out on the surface of polished sections of pyrite by heating to 600° C. for many hours. He concludes that the gold is not combined with sulfur but is contained in the lattice of pyrite in molecular or atomic form. It seems doubtful whether the relatively large gold atom, and much less colloidal particles, would fit into a tight lattice like that of pyrite. The opinion of the writer is that during the deposition of pyrite and other sulfides gold from colloidal solutions is adsorbed in various proportions on the sulfide surface but does not enter into the lattice of the mineral. Under normal conditions, the gold in the sol would be mainly deposited at the end of the period of mineralization. But, in part, it would be adsorbed on the various sulfides during their deposition. Regarding the many other factors that might cause coagulation of the gold sol may be mentioned escape of certain volatiles such as CO_2 and H_2S. Probably most important is the deposition of the silica, being the main stabilizer. When the bulk of the silica is deposited in crystalline or metacolloid form, the gold will of necessity also be precipitated.

To conclude, we do not know in what form the gold was contained in the emanations as they separated from the residual silicate magma. After the neutralization of the acid emanations it seems probable that the gold occurred as a sol stabilized by the presence of H_2S, Na_2S and silica sol. And it is also probable that it will persist in this form until the end of the deposition.

From all these incomplete data, the following tentative conclusions might be justified:

1. The presence of gold chloride as such in hydrothermal solutions is highly improbable.

2. The long-continued existence of double salts of alkaline sulfide and gold sulfide is highly improbable.

3. The only form in which gold can be present in hydrothermal solutions is as a sol, stabilized by silica and probably it was there in such condition from the highest to the lowest temperatures.

4. In the silica gels gold may be precipitated in dendritic form (National, Nevada) and on the quartz formed from the gel metallic gold will commonly be precipitated.

5. The earlier sulfides may adsorb a small quantity of gold in such finely divided forms that it is not visible under the microscope, nor amenable to amalgamation. This may take place in any deposit but is decidedly marked in many of the epithermal ores.

6. No adequate chemical explanation has been proposed for the simultaneous deposition of gold and silver, as in electrum or in a telluride like sylvanite.

7. The deposition of gold and gold-silver tellurides is not yet fully elucidated.

Summary

The temperatures of deposition of the minerals in igneous rocks, pegmatites and other magmatic deposits are briefly considered. In general these temperatures are much lower than the respective melting points, above which no mineral can crystallize. Most minerals in igneous rocks crystallize between 870° and 600° C., the temperature generally decreasing with increasing silica. Ore minerals separate out early, in part, but some of the metallic substances are tenaciously held and persist into the residual magma. The primary pegmatite minerals form about 573°; contact-metamorphic deposits crystallize generally below 573° and rarely as high as 1100°. Hydrothermal veins and replacement deposits form generally below 573° and some may form below 100° C. The criteria of temperature are considered: inversion points, unmixing, pressure, color, crystal forms.

Following Bowen and Fenner, the condition of the solution is considered during the transition from an igneous melt to hydrothermal conditions. After the main crystallization of the silicates the residual magma consists mainly of water and volatiles, with much silica and various amounts of heavy metals. The reaction is acid and the separation is effected by a boiling process, the pressure increasing rapidly. The acid extracts are soon neutralized or made alkaline by contact with surrounding rocks, and ascend towards the surface, perhaps impelled by the gas pressure from below. The reactions with the country rock continue, and the deposition of minerals begins. When the solutions reach the realm of the underground meteoric water, further reaction takes place. In epithermal deposits the solutions are often largely mixed with such waters.

The paragenesis or succession of minerals is next considered and a table presented showing the usual order of deposition. It is ascertained that this succession is in the main the same in hypothermal, mesothermal

and epithermal deposits. Local reversals and repetitions may occur, by various causes and agencies, but it is clear that the normal succession is caused by more powerful influences. Probably the temperature is the most important factor.

Colloidal solutions—sols—are most important in these dilute solutions and are generally produced by the reducing influences of H_2S, Na_2S, Te_2S and Se_2S. It is held that the silica and the tin dioxide are present as colloidal solutions, throughout the deposition, and that when coagulated the gels crystallize, more rapidly at higher temperature, more slowly as the solutions approach the surface. It is held that practically all sulfides, including pyrite, are first precipitated as gels. In colloidal solutions, the transportation problem of the sulfides is easy and comparatively rapid deposition can be effected.

After the normal sequence of sulfides and arsenides is formed, the lead-silver antimonides and arsenides, the argentite, the tellurides, the selenides and the native metals are deposited, the gold and tellurides accompanied by very small amounts of galena, chalcopyrite, etc. Parts of the gold and silver may be carried down with the earlier sulfides. It is held that this is caused by adsorption by the sulfides of a certain part of the gold and silver sols. In some epithermal pyritic deposits, the sulfides contain much invisible gold, precipitated, it is held, by such adsorption. The conditions causing this are not fully understood.

For silver sulfide an important inversion point is established at 180°, above which it forms argentite, below acanthite. It is believed that the silver sulfide was deposited from sols, stabilized by Na_2S or SiO_2. The occurrence of tellurium and selenium in volcanic sulfur shows that these elements are present with magmatic emanations. After neutralization of the acid extracts, H_2Te and H_2Se precipitate many tellurides and selenides, and are no doubt as potent as H_2S to transform these into sols. Calaverite has an inversion point at 184° C.; some deposits contain this mineral deposited above 184° C., others the same mineral with different structure formed below 184° C. Similarly, it is shown that hessite, the silver telluride, was deposited between 150° and 184° C. It is held that the gold occurs in hydrothermal solutions as a metal sol, stabilized by silica and that it occurs as such from the highest to the lowest temperatures. From this sol it is gradually precipitated by adsorption by the sulfides, but the larger part was held to the close of the deposition. Among the many other causes that might lead to the coagulation of the gold sol are escaping H_2S and CO_2 but the most important cause is probably the coagulation of the silica sol, which would be likely to be followed by the precipitation of the gold.

The presence of gold chloride in hydrothermal solutions is highly improbable, as is the long-continued existence of double salts of alkaline sulfide and gold sulfide. No adequate explanation has been presented

for the simultaneous deposition of gold and silver as in electrum or in sylvanite.

Melting and Inversion Points

Melting Points of Silicates

Degrees Centigrade

Orthoclase	1170
Albite	1122
Anorthite	1550
Nepheline	1526
Leucite	1686
Augite	1185 (variable)
Diopside	1391
Enstatite	1557
Aegirine	990
Biotite	1155 (variable)
Wollastonite	1540
Olivine (Forsterite)	1890 (Bowen)

Melting Points of Double Sulfides of Alkalis (Freeman)

Degrees Centigrade

$PbS.Na_2S$	650
$FeS.Na_2S$	660
$ZnS.Na_2S$	620
$Cu_2S.Na_2S$	560

Useful Inversion Points

Degrees Centigrade

Pseudowollastonite only above	1180
Quartz β above	573
Quartz α below	573
Quartz β only below	870
Tridymite above	870
Cristobalite above	1470
Sphalerite only below	1020
Wurtzite above but metastable below	1020
Ag_2S (argentite) isometric above	180
Ag_2S (acanthite) rhombic below	180
Hessite probably isometric above, probably rhombic below	150
Pyrargyrite and proustite form solid solution above, unmix below	473
Chalcocite, isometric above	91
rhombic below	91

Melting Points of Metals, Sulfides, and Oxides

Degrees Centigrade

S	119
Te	450
Se	217
Bi	271
Sb	630
Ag	960
Au	1063
Cu	1083
Realgar	310
Calaverite	472
Pyrargyrite	483
Miargyrite	509
Stibnite	546
Cinnabar	580 (sublimes)
Bi_2Te_2	600
Jamesonite	609
Argentite / Acanthite	843
Hessite ?	955
Niccolite	968
Galena	1120
Molybdenite	1185
Chalcocite (isometric)	1130
FeS	1193
FeS_2 (pyrite)	1171
Pyrrhotite	1157–1187
ZnS	1650
Cassiterite	1127
SiO_2 (cristobalite)	1713 (gas thermometer scale)
Magnetite	1538
Mn_3O_4	1705
Chromite	2180
Alumina	2250
Zircon	2550
ZrO_2	2700
Apatite	1285

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Geological Relations of Some Major Gold Deposits of the Canadian Shield

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(New York Meeting, February, 1937)

Gold occurs in many mineral deposits in the rocks of the Canadian Shield. It is present in the ores of many base metals and a considerable quantity is recovered as a by-product from the production of copper at Noranda, of nickel and copper at Sudbury and of copper and zinc at Flinflon. But the greater part of the gold production of Canada is from deposits in which gold is the only metal of value. No gold is obtained from placer deposits within the borders of the Canadian Shield.

A brief review of the geological relations of some of the gold deposits of the Canadian Shield will be presented in an attempt to explain certain associations. Only those which are being worked for their gold content will be considered and descriptions will be confined to representative types of deposits, chiefly those of major importance; a few others, differing in certain features from those of the older and larger mines, will be included in order to illustrate the general conditions in which gold occurs in these ancient rocks. Brief summaries will be given of the geological relations of the Beattie and Siscoe mines of northern Quebec, of the mines of the Porcupine and Kirkland Lake areas of northeastern Ontario and of the Howey, Central Patricia and Little Long Lac mines of northwestern Ontario.

Genetic relations to igneous rocks of granitic composition, outcropping in the vicinity, have been assumed for many of these deposits. Undoubtedly the ore minerals have been derived from igneous sources and, for some occurrences, there is evidence suggestive of genetic relationship between the ore bodies and neighbouring intrusive rocks.

During the past few years, however, many investigators have expressed doubt of the genetic relationship of certain ore bodies to igneous masses near which they occur. As will be seen from references in the following descriptions the importance of the structural conditions in localizing deposition has been realized. This is pointed out for Canadian pre-Cambrian gold deposits by E. T. Dougherty, who also refers to the

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common assumption that gold deposits are related to certain types of intrusives. He suggests the need of much further critical study and analysis of the relations[1].

In many cases evidence seems to indicate that the proximity of ore bodies to exposed igneous rocks is due to structural factors only and has no genetic significance.

Geological Relations

The geological successions in the vicinity of many of these deposits are similar in some respects, but differ so materially in others that exact correlation of the rocks of the various areas is impossible. Generally, rocks of three or more ages are present. The oldest are chiefly volcanics. A series made up mainly of clastic sediments, but with local flows or tuffaceous beds, lies unconformably on the oldest series. Intrusives of some variety and of different ages invade both volcanics and sediments. Although local names have been used frequently, it has become more and more the custom to apply the term "Keewatin" to the volcanic rocks, "Timiskaming" to the sediments and local volcanics that lie above the oldest series, and "Algoman" to the assemblages of rocks of granitic or granodioritic composition that intrude both these series. In many areas there are minor basic intrusives, and there are also granites earlier than the Timiskaming, since pebbles of granite are found in the conglomerate of that series. None, however, can be recognized in place in most regions. The inexactitude of the common use of these terms is evident. The rocks referred to as Keewatin are mainly volcanics, those grouped as Timiskaming are terrestrial sediments and those called Algoman are intrusives. None of these could be expected to extend for great distances. There is no age significance in the fact that the oldest series of rocks, associated with the deposits chosen for discussion, consists of volcanics. In the Grenville area of southern Ontario, where less important gold deposits are found, the oldest rocks are chiefly sediments among which limestones are important members. The use of the terms Keewatin, Timiskaming and Algoman should not be considered to imply even approximate correlation between the rocks of widely separated portions of the Canadian Shield.

Gold-bearing veins occur in the ancient volcanics, in the sediments that overlie them, and, in some places, in the post-sedimentary intrusives.

Gold Deposits in Western Quebec

The Siscoe mine[2] is on Siscoe Island, Lake Montigny, Quebec Province (Fig. 1), near the western extremity of a batholith of granodiorite, the surface diameter of which is 15 miles. The batholith intrudes lavas,

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[1] References are at the end of the paper.

which are classed as Keewatin (Fig. 2). The granodiorite has been fractured and the fractures filled either by dikes or by gold-bearing veins. Some gold occurs in tourmalinized wall rocks but most of it is present in crushed zones in quartz, tourmaline or pyrite. The veins are clearly

FIG. 1.—KEY MAP SHOWING LOCATION OF DEPOSITS DESCRIBED.

later than the consolidation of the granodiorite; the veins and the dikes that cut the granodiorite are closely associated and may have a common magmatic source.

The Beattie gold mine[3,12] is in the northwestern part of the Province of Quebec (Fig. 1). The ore body is large, but of low grade. Rocks in the vicinity of the mine are: (1) pillow lavas and tuffs, which are assigned to the Keewatin series,[3,12] (2) greywacke, classed as Timiskaming, and

(3) syenite porphyry and bostonite porphyry, which intrude the Keewatin rocks and are probably younger than the Timiskaming sediments. The

FIG. 2.—GEOLOGICAL MAP OF SISCOE AREA, QUEBEC PROVINCE. (*After J. E. Hawley*[2].)

bostonite porphyry is younger than the syenite porphyry. It was intruded following or accompanying shearing in the syenite porphyry and commonly followed the zones of shearing.

After intrusion, the bostonite porphyry was brecciated in a zone adjacent to the contact with the earlier porphyry (Fig. 3). Subsequently

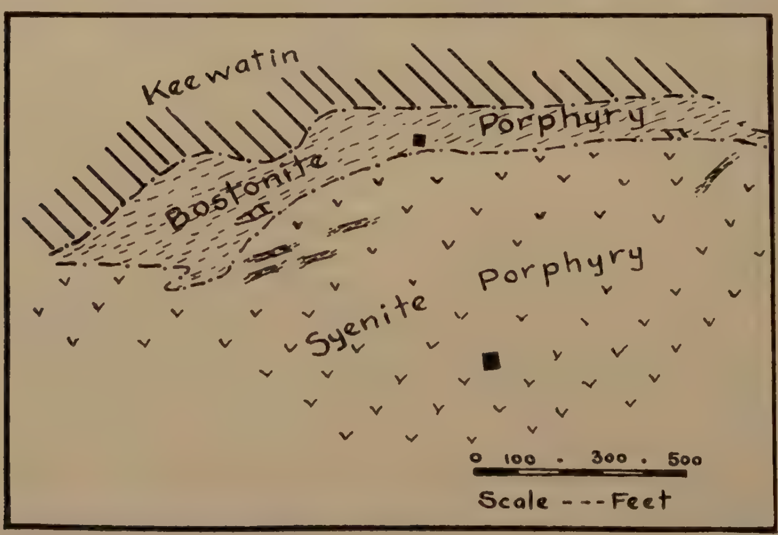

FIG. 3.—SURFACE GEOLOGICAL MAP OF BEATTIE MINE. (*After J. J. O'Neill*[12].)

the fragments were recemented and partly replaced by quartz. Later there was further replacement by sericite and carbonates and finally

pyrite, arsenopyrite and gold were introduced, in extremely small grains and crystals.

All of the metallic minerals seem to have been formed at approximately the same period. Clearly the solutions from which they were deposited were much later than the consolidation of either porphyry and the source of the solutions must have been at some considerable distance from the parts of the ore bodies now exposed, since the minerals present in the ore are typical of moderate temperatures and pressures.

L. V. Bell and A. M. Bell have discussed in considerable detail the structural features of several other gold deposits of western Quebec[4]. They stress the structural control of gold deposition and conclude that:

> the deposits are genetically related to the intrusive masses in which they occur, a more specific control of mineral deposition is produced by structural factors, notably fracturing under certain limiting conditions.

Gold Deposits of the Kirkland Lake Area[5,6]

The Kirkland Lake area is in the eastern part of the Timiskaming district of the Province of Ontario (Fig. 1). Here are situated half a dozen large, productive gold mines, including Lake Shore, Teck Hughes, Wright Hargreaves, Sylvanite, Kirkland Lake, Macassa and Tobourn.

The geological formations in the area are assigned to the Keewatin, Timiskaming, Algoman, and possibly Keweenawan periods. All the rocks classed as Keewatin are flows. They do not occur near the veins. The rocks placed in the Timiskaming series are conglomerate, greywacke and a considerable thickness of tuffs. The Algoman rocks consist of basic syenite (lamprophyre), syenite and syenite porphyry. They were intruded along the axis of a syncline of Timiskaming rocks and now form a broad band of heterogeneous character which separates the Timiskaming sediments of the north limb of the syncline from those of the south limb. Lastly, a few diabase dikes were intruded following north-south directions transverse to the trend of the older rocks (Fig. 4).

The veins occur in east-west fault zones. In the central part of the area there are two main vein zones, of which the northern is generally the stronger and contains the more productive ore bodies. It dips southward at an angle of 85°. The movement along the fault in which the vein lies has displaced the south side upward and to the east. Todd has estimated that the vertical component of this thrust is approximately 2000 ft. and the horizontal component about 550 ft. (ref. 5, 43–47).

The south zone is 400 ft. south of the north zone and, although shearing along it has been somewhat less intense, important bodies of ore occur in it. Smaller, parallel ore bodies lie near both main zones, and diagonal fractures linking them also contain ore in places. The ore bodies are lodes formed by the replacement and impregnation of crushed and brecciated rock along the zones of movement.

FIG. 4.—GEOLOGY OF PART OF KIRKLAND LAKE AREA. (*After E. W. Todd*[5].)

The rock of the vein zones, chiefly syenite and syenite porphyry, is brecciated and much altered. Stringers of quartz introduced into the breccia are themselves fractured. In the later fractures more quartz, some carbonates, pyrite, chalcopyrite, galena, tellurides and gold were deposited.

Syenite porphyry and syenite were apparently more brittle than basic syenite and hence more favorable for the development of wide and continuous zones of fracturing. The most favorable conditions, however, seem to have been an intimate interfingering of syenite or syenite porphyry and basic syenite.

Gold Deposits of the Porcupine Area[7]

Gold was discovered in the Porcupine area, 50 miles northwest of Kirkland Lake, in 1909. Production began two years later. The large producing mines are Hollinger Consolidated, McIntyre-Porcupine, Dome, Coniarum and Buffalo-Ankerite. The Pamour mine, the ore bodies of which may be very extensive, is now being developed.

In this area Keewatin volcanics are overlain unconformably by Timiskaming conglomerate and greywacke. Both series are closely folded so that the sediments form a syncline which plunges to the northeast. Probably after the development of this structure, the folded rocks were intruded by quartz porphyry, the bodies of which are pipelike, with elliptical cross sections, and with steep rakes to the northeast. Granite occurs some distance both to the south and to the north of the gold-producing area. It is intrusive into the lavas, but is not in contact with other rocks. After the porphyry was consolidated, considerably altered, and sheared, ore-bearing solutions entered the fractures. The wall rocks were altered and veins were formed containing some ankerite or calcite, tourmaline, scheelite, clinozoisite, pyrite, chalcopyrite, gold and gold tellurides. Not all of these minerals are contemporaneous, but it seems probable that they were formed in a definite sequence, during a single process accompanied by some shearing[8]. Deposition of gold took place during much of the period but was probably most abundant toward the latter part of it.

Certain geological features are significant. The intrusion of the quartz porphyry caused severe alteration of the volcanics with the production of carbonates. The vein-forming solutions also caused alteration of the walls of the veins, introducing more carbonate and producing a considerable amount of sericitization and pyritization. The veins occur in the lavas, in the sediments and in the quartz porphyry, but, in general, those in the porphyry are erratic in their gold content and only the vein itself carries gold in sufficient quantity to form ore. On the other hand, where the host rock is greenstone or greywacke, a considerable amount of gold occurs in the altered wall rock, especially where there is much pyrite:

as much as a third of the ore mined from such bodies consists of material other than the vein filling proper. Individual veins are lenticular and ore bodies are of the nature of lodes with lenses of ore closely spaced.

In the central, and so far most productive part of the area, ore is confined to a zone within 1000 ft. of porphyry bodies (Fig. 5). At the Pamour mine, however, no porphyry has yet been found and the ore zone lies along the contact between greywacke and conglomerate of the Timiskamian series, and volcanics of the Keewatin.

Fig. 5.—Porphyry bodies and vein systems of Hollinger mine, from mine plans of Hollinger mine.

Gold Deposits of Northwestern Ontario

Several mines in the region north and west of Lake Superior have begun to produce gold during the past few years. The amount coming from them is small as yet, compared with that from the mines at Porcupine and Kirkland Lake. The geological conditions under which the gold occurs is somewhat different from that in the eastern deposits.

The Little Long Lac mine is situated 50 miles north of Lake Superior, and the same distance east of Lake Nipigon. Gold was discovered in 1932 and the mine is already an important producer. The consolidated rocks of the area are lavas, overlain unconformably by a sedimentary series composed of conglomerate, greywacke, slate and iron formation. The volcanics are tentatively assigned to the Keewatin series, the sedi-

ments to the Timiskaming. Rocks of both series are cut by dikes and small masses of diorite and of feldspar porphyry. The latest consolidated rock is diabase, which forms small north-south dikes. The Timiskaming formations lie in a syncline that plunges westward at an angle of 40°. At the Little Long Lac mine the rocks are sediments, the beds of which are nearly vertical except where they are involved in complicated drag folds. There is a nearly east-west foliation.

The gold-bearing veins lie in east-west shear zones. The widths of individual veins vary commonly from a fraction of an inch to 6 or 8 in., but veins are closely spaced. The vein minerals are fine-grained quartz and small amounts of pyrite, arsenopyrite, chalcopyrite, bournonite, stibnite and gold. The wall rocks contain little gold and show but little alteration by the vein-forming solutions. Some small intrusive bodies of diorite occur not far from the vein zone, but near the veins there is no outcrop of igneous rock of a kind that could have been genetically related to the magma responsible for the ore-bearing solutions.

The shear planes in which the veins occur seem to be localized near the crest of a drag fold: hence they cross the bedding planes at large angles. The ore shoots are confined to a bed of massive arkose and end where the fracture zone enters the beds of greywacke above and below the arkose. Thus the ore shoots rake westerly with the plunge of the folds.

Northwest of Lake Superior there are several mines from which gold is now being obtained. The ore bodies of two of these will be described, because they differ somewhat from those previously considered.

The Howey mine is on the south shore of Red Lake, 50 miles east of the boundary between the provinces of Ontario and Manitoba (Fig. 1). The rocks outcropping near the ore bodies are: (1) greenstone and associated schists, (2) quartz-porphyry dikes intruding the greenstone, and (3) granite. Granite does not occur in the vein zone, but outcrops as a boss 5 miles in diameter a short distance northwest of the mine.

At the Howey mine, a quartz-porphyry dike has suffered considerable fracturing. The fractures are filled with quartz containing small quantities of the metallic minerals—pyrite, sphalerite, chalcopyrite, altaite and gold. The quartz porphyry forming the walls of these veins contains very little gold, but the veins, although lenticular and only a few inches wide, are so closely spaced that parts of the dike can be profitably mined.

North of Red Lake two other mines, the Central Patricia and Pickle Crow, are now producing gold[9,10]. At the Central Patricia mine (Fig. 1) the rocks in the vicinity of the vein are basic lavas, iron formation and a small dike of syenite (?). The ore bodies are lenticular and consist of mineralized parts of a fractured zone in the iron formation, which dips 75° northward. The ore lenses rake east at an angle of approximately 60°. The ore consists in part of minerals filling fractures, in part of

minerals that have replaced constituents of the iron formation. The chief metallic minerals present are pyrrhotite and arsenopyrite. Chalcopyrite and pyrite are less abundant. Gold is probably in metallic form but in extremely small particles, as it is rarely visible even under high magnification. It seems to be associated with all of the sulphides except pyrite. Nonmetallic minerals in the ore are quartz, carbonates and chlorite. A relationship of the shear zones in the iron formation to a fault 350 ft. north of the shaft, and the control of the rake of the ore bodies by the intersection of the shear planes and joint planes, have been suggested[11].

General Relations

Mineralogy.—Most of the gold deposits of the Canadian Shield consist of lenticular veins in which quartz is by far the most abundant mineral. In some veins there are small quantities of feldspar, tourmaline, clinozoisite, scheelite or zoisite, and, in a few, considerable proportions of ankerite or calcite. Pyrite is the most abundant metallic mineral and occurs in nearly all deposits. Arsenopyrite is abundant in some; in many it is lacking. Commonly there are small quantities of chalcopyrite, sphalerite or galena. Tetrahedrite, bournonite and berthierite have been found, but the sulphosalts are present sparingly. Tellurides of several varieties are present in many auriferous veins, but never in large amounts. Varieties reported are sylvanite, calaverite, petzite, altaite, coloradoite and tetradymite. Most of the gold is present as the metal and commonly is most abundant along dark streaks in the quartz, which contain sericite and chlorite. Often pyrite is auriferous. Microscopic examination usually shows that it contains gold in tiny particles, but there is a possibility that some is in solid solution with the pyrite.

Wall-rock Alteration and Structural Features.—The wall rocks of the veins have been altered by the vein-forming solutions to different degrees, possibly because of differences in the kind of rock in which the veins occur. Basic lavas such as those in the Porcupine area were already much altered before the period of vein formation. At some distance from the veins, analyses show that they contain quartz 10 to 25 per cent, chlorite 20 to 25 per cent, carbonates 10 to 15 per cent, plagioclase 25 to 30 per cent, sericite 2 to 8 per cent, and varying amounts of clay minerals. The rocks adjacent to veins contain a much larger proportion of sericite and usually a large amount of pyrite, some of which seems to have developed from chlorite. Where the wall rocks of gold veins are of granitic composition, as are those of the Kirkland Lake deposits, alteration produced along the veins is much more easily recognizable, although probably not more severe than in basic wall rocks. Commonly sericite is present in abundance. Much of it is the result of hydrothermal alteration of the feldspars. Quartz is in much larger quantity than in the fresh rocks. Other changes

are not quantitatively very important. Wall-rock alteration in veins in sedimentary rocks is similar to one of these types, depending upon whether the rocks contain much feldspar or are chiefly chloritic.

Structurally the deposits vary considerably, but generally they are related to shear zones. Most individual veins are short and lenticular, but sufficiently closely spaced to form lodes. In a few deposits, single veins extend for considerable distances.

Relationship to Igneous Rocks.—The fact that igneous rocks, of granitic composition, outcrop in the vicinity of most of the gold-bearing veins of the Canadian Shield has been accepted rather generally as evidence that the vein material and the intrusive rocks are genetically related. The relationship of quartz veins to some magma from which granitic rocks could be derived seems undeniable, and, in some of the gold occurrences described, the minerals present support that hypothesis. The veins found in the Siscoe area, for example, have a characteristic association of gold with tourmaline, an association that has not been found far from the granodiorite masses in which the productive veins occur. Should it be found that this characteristic mineral assemblage actually is confined to the vicinity of these intrusives, the derivation of the gold deposits and of the granodiorite from a common magma seems a reasonable hypothesis. In that case the granodiorite was sufficiently consolidated to be fractured in the outer or upper part of the batholith, while deeper portions of the parent magma were still fluid; the fractures were filled by the uprising vein solutions presumably given off in late stages of the cooling of the magma.

The ore body at the Beattie mine may be explained by a similar process involving a prolonged period of igneous activity of which the syenite porphyry, bostonite porphyry, the silicification and the pyrite-arsenopyrite-gold deposition are successive facies.

The relationship of some other gold deposits to intrusive rocks outcropping in the vicinity is less easy to establish, and occasionally it seems clear that there is no such genetic relationship. At Kirkland Lake the post-Timiskamian intrusive rocks, in order of age from oldest to youngest, are basic syenite (lamprophyre), syenite, syenite porphyry and diabase. The three syenitic types are believed by Todd to be differentiates of a common magma[5]. The diabase dikes, however, have a north-south trend transverse to the trend of the syenitic rocks; lithologically the diabase is similar to that assigned to the Keweenawan in other parts of the region. As the dikes are displaced by the vein zone it seems difficult to postulate any logical sequence by which the ore minerals could have been derived from the magma from which the syenitic rocks were formed. At Porcupine the gold-bearing veins are younger than the quartz-porphyry bodies near which many of them occur, and no genetic relationship between the quartz porphyry and the ores seems possible. There are

no other acidic intrusives in the immediate vicinity of the gold deposits, to which they could be genetically related. Both at Kirkland Lake and at Porcupine, mining operations have now reached depths of approximately a mile beneath the surface. There has been no marked change in the character of the vein material. Hence it seems likely that the locus of the magmatic source of the ores must be still far beneath the present workings.

Structural Factor.—Where the veins have no genetic relationship to exposed igneous rocks, it is necessary to offer an explanation of their proximity to them. An important factor in localization of all veins is structural control of fracturing. Rocks that behave as brittle members under stress are fractured, and the fractures are later occupied by veins if mineral-bearing solutions are available and can enter the fractured zones. The physical character of a rock determines its behavior under stress; its origin is of no consequence. The most likely place for failure is near contacts between rocks of unlike competence. This condition of heterogeneity is supplied by various associations. At Porcupine the McIntyre-Hollinger vein system is in a block of ground characterized by a combination of lava flows and quartz-porphyry stocks. The latter probably acted as buttresses between which the block of lavas was sheared. At the Dome mine, sedimentary beds of somewhat different character, lava flows, and masses of quartz porphyry supply favorable conditions. At the Pamour mine, slipping between massive beds of conglomerate and the lava flows seems to have been the process by which fractures were formed. At Kirkland Lake, the zone of most abundant and intense fracturing is in a complex of intrusive rocks of unlike brittleness. At the Little Long Lac mine, a thick, massive bed of arkose was sufficiently brittle to be fractured; the ore bodies are confined to that horizon. Iron formation, interstratified with lavas, is the favorable member at the Central Patricia mine. Many other deposits fall in one or other of these structural types.

Summary

Thus it appears that, although the ore-bearing solutions were of igneous origin, there are few places in which their genetic relationship with intrusive rocks, outcropping in the vicinity, can be satisfactorily established. In some places where genetic relationships between ores and certain igneous rocks have been assumed, later investigation has shown that there is a wide diversity in age. In deposits such as those of Porcupine or Kirkland Lake, it seems likely that the solutions that formed the veins were derived from a magma far below the present surface, that the veins are localized by structural factors, and that it is doubtful whether any large outcrop of igneous rocks, genetically related to the ores, occurs near them. The fact that contacts between igneous rocks

and those that they intrude are structurally favorable for the development of zones of fracturing, in some deposits at least, explains the occurrence of gold-bearing veins near intrusive bodies.

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Gold Deposition in the Black Hills of South Dakota and Wyoming

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(New York Meeting, February, 1936)

The occurrence of gold, gold-silver, silver-lead-zinc ores in the post-Cambrian sediments in the Black Hills of South Dakota, and their genetic relationship to the Tertiary intrusives, is well known and accepted, but the occurrence of gold in the pre-Cambrian rocks at the Homestake mine, which underlie this Tertiary gold mineralization, has given rise to differences of opinion as to the period of major gold introduction into the older rocks. Fig. 1 illustrates the distribution of mineralization along the axis of the Tertiary uplift in both pre-Cambrian and post-Cambrian rocks. The following possibilities have been presented:

1. The Homestake gold deposit was formed in pre-Cambrian time and the deposits in the overlying Cambrian sediments formed during Tertiary time.
2. The Homestake lode existed at the beginning of Cambrian time, but was enriched by the passage of Tertiary auriferous solutions.
3. The Homestake lode was represented by pre-Cambrian quartz-chlorite sulfide mineralization, very low in gold content and the gold superimposed on the older nearly barren mineralization during the Tertiary metallogenic epoch.
4. The sulfides and sulfarsenides as well as gold were deposited in both pre-Cambrian and post-Cambrian rocks during the Tertiary period.

Those favoring the first proposition feel that the question is pretty well settled, while the proponents of the second theory are more flexible in their view, feeling that the amount of gold introduced in either period is still an open question. The third and fourth theories, with some modifications, are supported by the proponents of the Tertiary theory, who feel that the evidence strongly favors that period.

The following pages are intended to present the evidence and to give the reaction of various observers with my own comments and conclusions; also, the accumulated facts of observation gathered over a period of close association with the area under discussion, beginning in 1919.

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The terms "Tertiary ore" or "Tertiary deposit" as used in this paper relate to mineralization in post-Cambrian rocks. The words "Tertiary

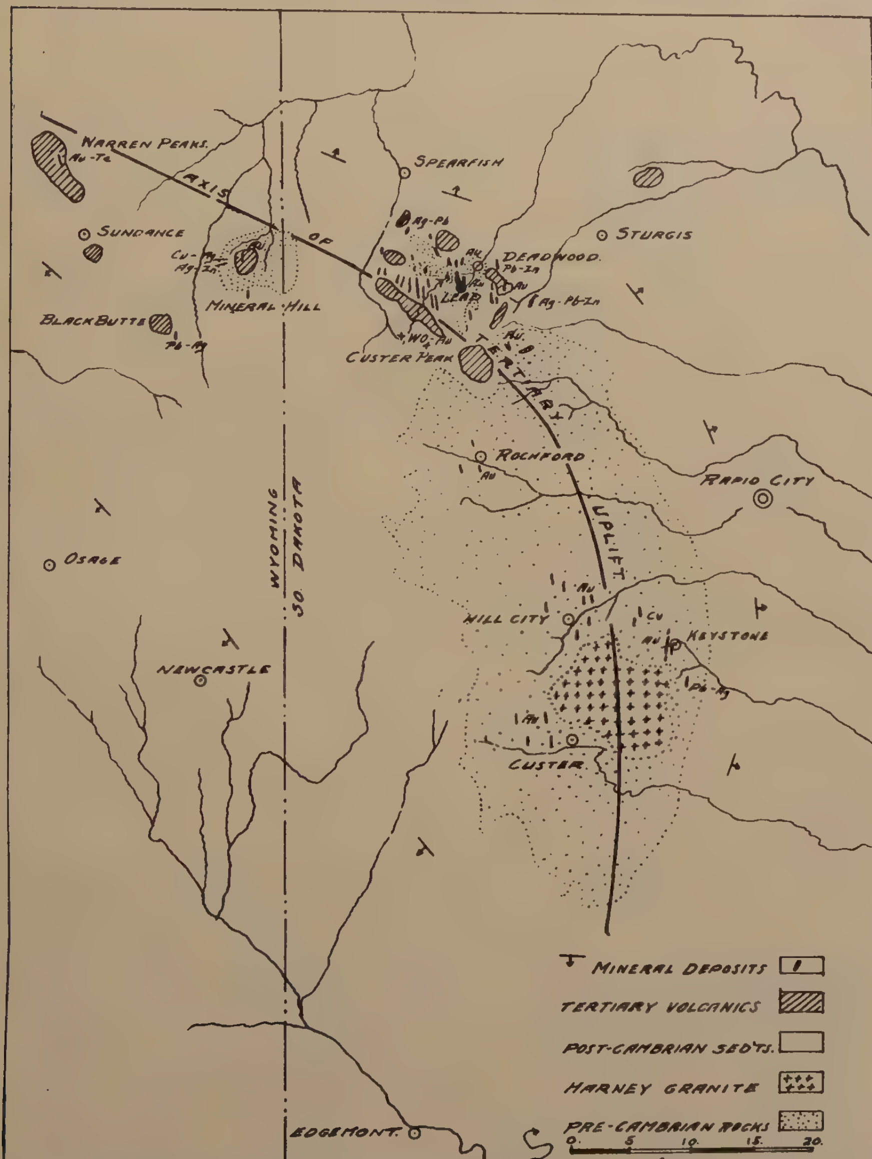

FIG. 1.—ARRANGEMENT OF MINERALIZATION WITH RESPECT TO AXIS OF UPLIFT AND IGNEOUS ROCKS, BLACK HILLS, SOUTH DAKOTA AND WYOMING.

mineralization" have not been used as applying to the Homestake ores except where specifically so used, but it is assumed during the discussion

that the term "pre-Cambrian" applies to ore as well as rocks unless otherwise specifically stated.

General Geology

In general, the Black Hills form an oval uplift approximately 100 miles long and 40 miles wide across the central portion. At the northern end an additional area of uplift is consequent upon an area of Tertiary intrusives embracing about 100 sq. miles. This area has produced the bulk of the gold. Adjacent to the Harney micropegmatite and granitic masses of the Central Hills, which are intrusive into the pre-Cambrian schists, are a number of smaller ore deposits. The ratio of gold production from the deposits adjacent to the pre-Cambrian Harney granitic area to that of the Northern Hills Tertiary intrusive area is close to 1.0 to 100.0. There are 27 miles of intervening pre-Cambrian outcrops containing quartz veins, shear zones and sulfide replacement bodies in which the value per ton is relatively low or, if not, the quantity is small when considering present commercial ranges. .

That the entire uplift is underlain by a huge Tertiary batholith is evidenced by the flanking sediments in which, from the conglomerate at the base of the Cambrian dolomitic rocks to the Oligocene deposits, is embraced almost the entire geologic column. Magnificent inward-facing, outward-dipping escarpments of massive Pahasapa limestone, Dakota sandstone, etc., give silently impressive testimony to the easternmost thrust of the Cordilleran effort. At the southern tip of the Hills, the Cretaceous sediments are tilted to angles as steep as 45° from the horizontal, while along the western rim, near Newcastle, Wyo., they are as steep as 65°. The northern and eastern slopes, while definitely tilted, are more moderate in inclination.

The Tertiary intrusives at the northern end of the uplift are, in the main, acid rocks, ranging in composition and classification from phonolites to rhyolites, with almost every conceivable gradation in both texture and composition between. Their mode of intrusion is also variable. Dikes following the schistosity of the pre-Cambrian complex, when encountering the nearly flat, overlying sediments, have found lines of least resistance in the horizontal bedding planes and hence become sills, often laccolithic in disposition. Later dikes, but related to the same period of volcanism, are seen cutting through the whole. Small dikelets of intrusive breccia are found as the last evidence of activity. The mineralizing solutions evidently have arisen prior to the intrusive breccias but subsequent to the rhyolites in some instances and following the breccia in others. (The intrusive breccia in the Big Vertical at Annie Creek mine, according to Dr. Kerr, Columbia, has been subjected to intense silicification and mineralization.)

It is obvious, then, from these facts and broad field evidence, that in the Northern Black Hills auriferous solutions originating from the Tertiary intrusive magma arose through channelways in the pre-Cambrian rocks and finally reached the Cambrian and later sediments, overlying, while in the Central Hills ore deposits are probably related to the Harney granite.

Geologic Investigations

The first geologic work in this section was done by Dr. Hayden in 1854. He was commissioned by the government to investigate the plains area. Reaching the Black Hills, he prepared a map, which, considering the knowledge and means available at the time, was essentially accurate as to the major structural features. He recognized the presence of inward-facing escarpments of sediments, dipping away from the general center of the uplift, as a series of younger rocks exposed by the truncation of the uplifted dome. As Dr. Hayden's commission was mainly a study of the lignite areas, and as gold was probably not discovered at that date, no mention of it is known to have been given.

In 1874 the next period of interest commenced. White prospectors came in increasing numbers into the Black Hills, which had been set aside as an Indian Reserve. Troops were sent to protect the rights of the Indians and in so doing turned back many prospecting expeditions. In the next year it became evident that the lure of gold and the persistency of the prospector was to assume command and the mining industry began to take possession.

G. C. Hewett[1], W. B. Devereux[2], J. D. Irving and S. F. Emmons[3], Darton and Paige[4,5] have been the principal contributors to the general knowledge of Black Hills geology. G. C. Hewett, as early as 1903, concluded that the Tertiary intrusives were responsible for gold deposition of the Northern Black Hills and expressed the opinion that because of the great thickness and large extent of the Tertiary laccolith the Homestake gold ore was also Tertiary in age. Irving and Emmons deal more particularly with the economic geology of the Northern Hills, having made a detailed study of the Tertiary mineralization of Cambrian and later sediments. W. H. Emmons[6] has published papers on the ore deposits genetically related to the Central Hills pre-Cambrian intrusives, as well as pointing out the zonal distribution of Tertiary metals around the Homestake tungsten-gold "hot center."

In dealing with the mineral deposits of the Northern Black Hills, Emmons says[6]:

> In most of these districts the ore deposits are closely associated with andesites, dacites and shallow-seated porphyries. Only rarely are typical zones developed, although there are few, if any, important reversals of the normal series passing downward in individual veins. In many districts the metallization is concentrated in

[1] References are at the end of the paper.

areas 3 or 4 m. square. The inference that these deposits lie above high points of unexposed batholiths is pure speculation, yet it is supported by many facts, particularly by their geological and geographic surroundings, and by the group of hypotheses developed from the study of mining districts where erosion has gone only slightly deeper than in areas containing deposits of group 1.

In a few districts dikes are closely spaced in certain centers suggesting that a high point of a deep-seated intrusive lies below, and rude zonal arrangements in the normal order are shown around the areas of closely spaced dikes. Examples include the Northern Black Hills, Pine Creek district and Battle Mountain district in Nevada.

The deposits of the Northern Black Hills are believed to form a cryptobatholithic system. The metallized area is made up of pre-Cambrian schists surrounded by sedimentary rocks that dip away from the schists. Porphyries, probably early Tertiary, intrude both the pre-Cambrian and later rocks. The larger number of the deposits probably were formed at about the time of the intrusion of the porphyries and it is believed that the metalliferous solutions were derived from a deeper seated igneous mass that supplied the porphyries. The most valuable deposits are those of the Homestake mine which replace folded calcareous beds in the pre-Cambrian schists. Irving and Paige concluded that the deposits were formed in pre-Cambrian time and this theory has long been accepted, but Hosted and Wright have stated their belief that the deposits are Tertiary, for which they find support in the close association of the ores with the rhyolitic prophyry dikes of Tertiary age.

In 1921 Hosted and Wright[7] reported, in part, as follows:

The pre-Cambrian rocks have been dynamically metamorphosed, making the group a series of plunging, tilted folds. The axes of the folds strike N. 60. W., to N. 35 E., and the axial planes dip steeply to the East with but few local exceptions. The plunge to the folds is to the South-east. The noses of the anticlines and the troughs of the synclines are highly contorted, fractured and squeezed, and were ideal places for mineralizing solutions to penetrate and replace.

. . . .

The ore fades away from the noses of the anticlines, leaving the long straight limbs of the folds barren or nearly so, therefore folding was one of the main factors controlling deposition. The rhyolite porphyry intrusions cutting through folded calcareous beds were accompanied or immediately followed by mineral bearing solutions which followed closely the course taken by the intrusives . . . .

In Fig. 2 is shown the distribution of ores in the Northern Hills in pre-Cambrian and post-Cambrian rocks, the association of ore and rhyolite dikes, and the trend of these as influenced by structure in the pre-Cambrian.

In 1923, Hosted and Wright published[8] the results of three years of investigation of the area embracing and surrounding the Homestake mine. In addition to definite conclusions concerning the periods of gold introduction, the intricate problem of major folding and its relationship to ore distribution was dealt with. Perhaps of greatest importance was the division of the pre-Cambrian into definite stratigraphic units.

Sidney Paige[9] had gone part way in this work in 1913, but ascribed the Homestake mineralization to a broad system of calcareous slates abutting a major fault. He recognized the complexity of the folding and

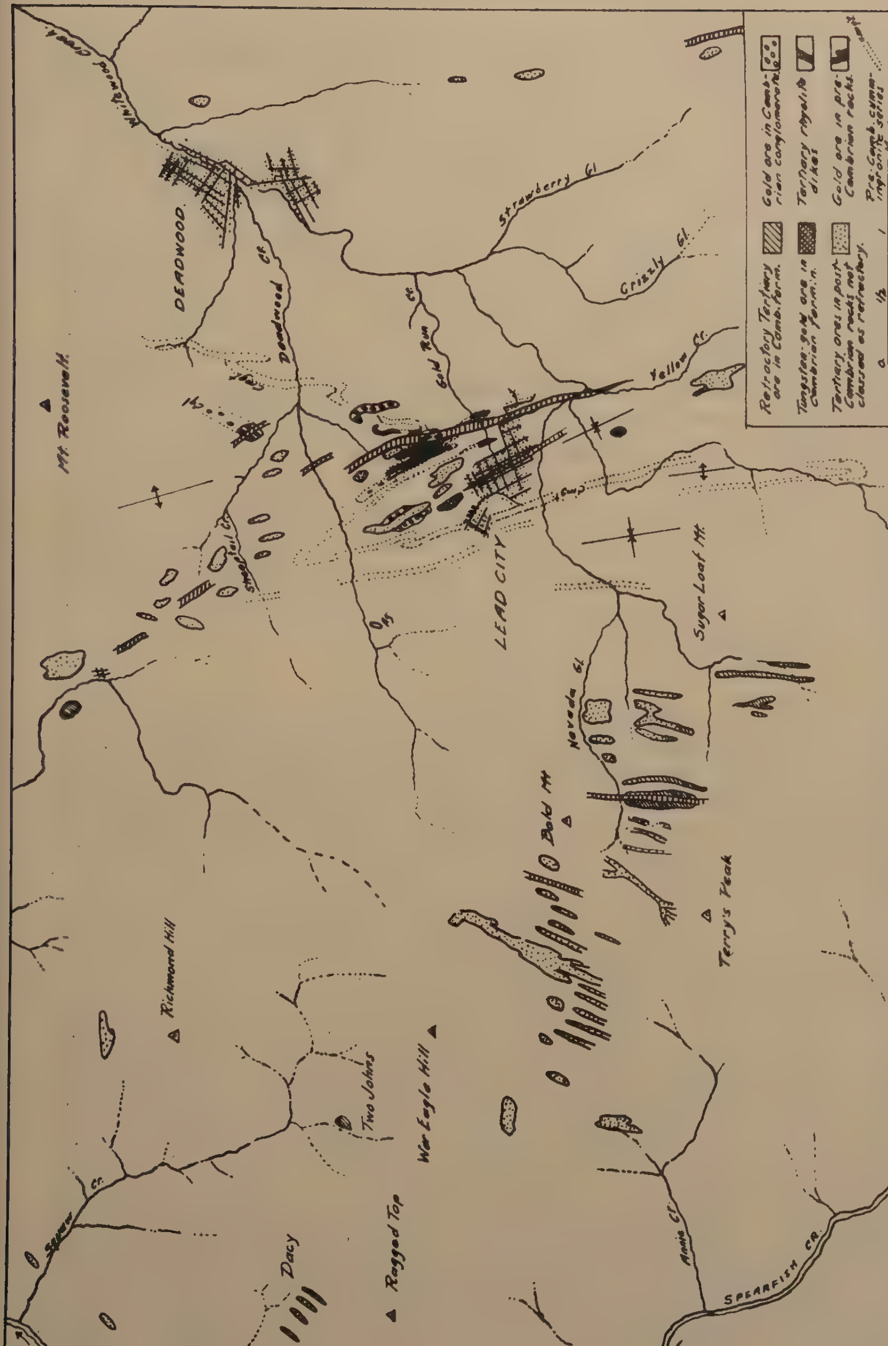

FIG. 2.—DISTRIBUTION OF PRINCIPAL ORE DEPOSITS, WITH TYPE OF REPLACEMENT INDICATED; ALSO, PRINCIPAL TERTIARY RHYOLITE ZONES AND DISTRIBUTION OF CUMMINGTONITE ROCKS (HOMESTAKE FORMATION).

suggested that the ore occurred in metamorphosed rocks that were derived from dolomites and that the lime content of these rocks often reached 50 per cent.

To Sidney Paige goes credit for giving the real key to the situation. At the time of his later visit in 1922, Hosted and I had divided the pre-Cambrian, segregating the cummingtonite series from the underlying siliceous rocks and the underlying calcareous rocks and terming it the "Homestake formation." We found that, as Paige indicated, the change to siliceous slates and quartzite at the top of the series was quite abrupt, but somewhat gradational at the bottom into the calcareous phyllites (Poorman formation). This gradational zone, typified by some cummingtonite and chlorite, less massive and definitely banded, we called the De Smet formation. Ore is found in these rocks but in general they are less amenable than the more massive beds of the Homestake formation.

At this time (1922), we did not agree with Paige's idea of the big fault, although the evidence he had at hand justified him in his early conclusion. We found that the abrupt change in strike of rocks adjacent to the large open cut was accomplished by tight folding. (See structure pattern, Fig. 2.)

In 1913 Paige did not mention ore genesis. On Nov. 21, 1922, he states, in a personal communication:

> I regard the arsenopyrite and pyrrhotite as distinctly pre-Cambrian sulfides, although there is also Tertiary arsenopyrite as noted by Irving. The age of the gold, however, is not a closed question yet. I think now there were two periods—one accompanying the pyrrhotite, another following the Tertiary volcanics. Devereux's and Irving's account of the fossil placers both suggest a pre-Cambrian lode, though the evidence is not conclusive. I feel that the Tertiary period of enrichment must have been important.

During the summer of 1923, Prof. E. S. Moore, Toronto, spent some time on the problem. He concluded[10]:

> My first conclusion supports that of Hosted and Wright regarding the Tertiary age of the ores in the pre-Cambrian rocks as well as those in the overlying Paleozoic strata. The close relationship of the intrusives to the ore bodies, illustrated by their proximity to one another and the influence of the impervious slates and quartzites in damming back solutions from intrusions, supports the view that the ore minerals came from the Tertiary intrusions. A specimen secured from the 1000 ft. level of the Pierce ore body illustrates very nicely the relations which can be observed at a number of points in the mine between the rhyolite and the ore. It shows that the ore in that section at least is later than the rhyolite as it has been injected into the rhyolite and it surrounds fragments of both brecciated rhyolite and schist. The presence of free gold and tellurides of gold in the ore bodies of both the pre-Cambrian and paleozoic formation favors a common origin and although the relative proportions of these minerals vary in different formation, this variation may be accounted for by differences in temperature of the depositing solutions and differences in composition of the wall rocks.

Moore also agrees with Paige on the order of deposition; i.e., arsenopyrite, pyrrhotite, pyrite, gold.

The correct interpretation of this problem of ore genesis became of extreme importance at this time for the reason that underground exploration, if it were to be carried on with good results, must be based on the best ore probabilities. Even though Lindgren[11] has classified the Homestake ore as pre-Cambrian owing to its occurrence in pre-Cambrian rocks and its association with quartz, chlorite, arsenopyrite, and Paige, in U.S.G.S. *Bulletin* 765, leaned toward the pre-Cambrian metallization as the most important (although modifying his view later, as quoted above), our observation that the gold itself seemed to be superimposed on an older less auriferous mineralization and was definitely distributed along the rhyolite (Tertiary) dike zone, seemed to me to be sufficiently clearly demonstrated as a physical fact to definitely confine such exploration to this zone.

The problem was studied further with the following major premises as guides:

1. Gold ores occur in pre-Cambrian and post-Cambrian rocks, therefore auriferous solutions must have transcended all of the rocks *subsequent to the formation of the pre-Cambrian quartz veins, sulfide bodies and shear zones.*

2. As pointed out by J. D. Irving, the Homestake ledges, being massive and harder than the adjacent rocks, occurred as a reef in the Cambrian sea. This is evidenced by the thinning of the basal Cambrian conglomerate in the vicinity of the Homestake lode. Therefore, if the lode was auriferous at the time of the formation of the conglomerate, gold in the channels and along the Cambrian beach line could be expected.

On this second point the evidence is quite positive. Irving states[3]:

> Some of the gold was dissolved by ferric sulfate resulting from the oxidation of the pyrite and from this solution was deposited in thin films in the schist below. This has produced an enrichment of the lowermost layers of the conglomerate.

Obviously, if the pyrite (Tertiary) entered into the enrichment of the lower layers of the conglomerate, these seams and nuggets must also have been formed in Tertiary time rather than at the time of deposition of the conglomerate, from his own argument and observations.

The distribution of gold in the conglomerate was confined mainly to three deposits, one of which Irving examined (Monitor workings, east of the Homestake lode), and noted that the mineralization here was accompanied by replacement of the calcareous matrix with values following seams of sulfides (oxidized), and the Phoenix-Minerva conglomerate mine in Blacktail Gulch, which was in a channel of conglomerate trending northeastward from the Homestake outcrops. These workings I examined in 1924, and found that the productive area lay adjacent to a

rhyolite dike which cut across the old channel at an angle of about 45°. The miners, with the placer origin idea, had drifted over 600 ft. farther down the bottom of the channel and prospected the rims without result. The pattern of productive stopes shows that the rhyolite was left between them as a pillar and the ore taken from rim to rim, either side of the dike, diagonally across the channel; obviously a replacement deposit.

The third and most spectacularly important occurrence of gold at the base of the Cambrian and near the Homestake lode occurred on the west side or "uphill" side of the lode. It was known as the Grants or Hidden Fortune discovery. Abundant free gold was taken, not only from the base of the unconformable contact, but *from above the quartzite bed that overlies the conglomerate horizon.* Many of the specimens were nugget-like, with rounded boundaries, which seem to represent the limit of mass growth rather than having been rounded by attrition. These ores graded into a siliceous tungsten-gold replacement of the dolomite horizon lying above the basal quartzite and obviously are replacement deposits. Even the trend of oreshoots conforms to the schistosity of the pre-Cambrian rocks below, which undoubtedly governed the strike of the several solution channelways.

In view of the above data, the theory of the erosion of a pre-Cambrian gold deposit did not seem to have much in support.

W. J. Sharwood (formerly chief chemist at Homestake) noted that the ratio of silver to gold in the Cambrian ores is greater than in the Homestake ore. He reports ratios of 5.0, 11.4 and 4.01 to 1.0 as against 0.215 to 1.0 and 0.433 to 1.0 for Homestake bullion[12]. His samples of Cambrian ore evidently were not large and came from widely separated workings around Lead City.

The difference in silver-gold ratio has been pointed out by advocates of the pre-Cambrian theory as showing that the solutions that precipitated the Homestake ore could not have been the same as those that precipitated the ore in the Cambrian and later rocks, owing to this and differences in reaction to metallurgical treatment. The work done by E. S. Leaver and J. A. Woolf[13] on the Cambrian formation ores, both "blue" and "brown" (unoxidized and oxidized), contributes some critical information on this point. Some comparisons are given in Table 1, based on their work, which involved the use of larger and more representative samples. These analyses show that there is even a wide variation in the ores wholly derived from the Tertiary solutions, but that there is a consistency in the "carrier" elements, arsenic and silica. This is also shown in Table 2.

From these and foregoing data it was further concluded that:

1. The consistency in the percentage of the "carriers," especially arsenic, is more than accidental and points strongly to a common origin for the mineralizing solutions.

2. That there can be a wide diversity between the ratios of the metals transported, especially considering samples from a radius of several miles; and that *the fact of such a variation does not argue separate ages of solutions.* In fact, some of the Tertiary mineral deposits in the district are predominantly lead-zinc-silver ores, but still containing arsenopyrite.

TABLE 1.—*Comparisons Based on Work of Leaver and Woolf*

	Composition, Per Cent				
	Au	Ag	SiO_2	S	As
Trojan brown ore	0.18 oz.	0.48	77.62	0.20	0.59
Trojan blue ore	0.23	0.51	70.20	3.68	0.31
Average ore	0.326	0.58			
Maitland blue ore (not representative sample)	0.98	8.51	72.70	8.93	0.33

TABLE 2.—*Comparison of Analyses*

	Composition, Per Cent				
	Au	Ag	SiO_2	S	As
Trojan ore (Leaver and Woolf)	1.0	3.07		3.24	0.33 (Tertiary)
Golden Reward ore (Leaver and Woolf)	1.0	1.26		6.50	0.38 (Tertiary)
Maitland ore (blue) (Leaver and Woolf)	1.0	1.41		4.54	0.47 (Tertiary)
Homestake ore, year 1918, mill run (Sharwood).	3.56	1.00		3.26	0.30 (pre-Cambrian and /or Tertiary)

3. That the difference in gold-silver ratio at different points in the district is less reliable by far as a key to origin of solutions than the percentages of introduced silica, arsenic and sulfur.

4. That the difference in metallurgy in various sections of the district has nothing in common with the origin of the gold (or other metals) but rather is accounted for by mode of replacement, which in turn is determined by local conditions. The host rock in the pre-Cambrian series is composed, mainly, of carbonates and metamorphic silicates, while that of the Cambrian was largely calcareous and subject to complete replacement.

5. That the Homestake host rock, being largely composed of ferromagnesian silicates, carbonates, quartz veins and barren sulfides when mineralized, would give rise to an ore very different in physical aspects from the host rock of Cambrian and later sediments, which was essentially dolomitic, unaltered and more finely crystalline. An intense silicification

could be expected in the latter, with sulfides locked in silica, causing the ores to respond reluctantly to amalgamation and cyanidation.

6. That the mineralizing solutions, under pressure and moderately high temperatures, filtered through the series of complex folds involving the cummingtonite series, deposited silica, arsenopyrite, pyrrhotite, pyrite and gold, the whole superimposed upon the pre-Cambrian quartz-chlorite mineralization, passing upward through the basal conglomerate, effecting here a pyrite-rich replacement and thence into the overlying dolomitic rocks, where further mineralization in important amounts was effected.

It was also deemed more than a coincidence that the average gold content of the Tertiary ores in the Cambrian rocks closely approximated that of the Homestake.

Still feeling confident that underground exploration, especially the trend of ore in depth, must be based on the assumption that ore would be confined to a definite zone paralleling the rhyolite dikes, Hosted and Wright in 1924 gave the opinion that certain outlying limbs of Homestake formation would, when dipping into the rhyolite zone in depth, become commercial ore.

These and other important predictions, based on the conclusions reached at that time, it is understood, have proved to be essentially correct.

The foregoing shows that the studies of all investigators up to 1924 recognized the presence, in varying degree, of pre-Cambrian mineralization, followed by further mineralization relating to the Tertiary volcanics. With one exception, all have admitted in one way or another that the gold introduced during the latter period was important, Irving giving excellent evidence favoring the replacement of the calcareous matrix of the basal conglomerate.

Factors Relating to Age of Gold Introduction

Does the fact that arsenopyrite, pyrrhotite and pyrite are the dominant sulfides in Homestake ore preclude the possibility of their being later mineralized? They are also prominent sulfides in the Tertiary ores. Kutz[14] even believes these sulfides and sulfarsenides were introduced in Tertiary time. He concludes:

> My microscopic study indicates that the opaque minerals in the Homestake mine were formed during one epoch of mineralization. This would seem that all of the gold is of the same age.
>
> The apparent lack of crushing and shearing of the sulfides and their similarity to minerals of the Deadwood lead and zinc mine suggest a Tertiary age of mineralization for the Homestake deposit.

Dwight E. Woodbridge, in a private report on the Maitland mine (in the Cambrian dolomites), dated 1906, writes:

They (Cambrian ores) are close grained and very hard and contain both pyrite and pyrrhotite.

A 50-lb. sample from the Portland mine, Bald Mountain district, gave by magnetic separation 1.29 per cent of pyrrhotite. A duplicate test gave 1.15 per cent pyrrhotite.

J. D. Irving[3] gives an analysis of ore from the Double Standard No. 4 shoot in which 4.10 per cent arsenopyrite is reported. Also, with the evidence of Leaver and Woolf[13] and Kutz[14] relative to arsenopyrite, it cannot be denied that the same minerals that are used as evidence of pre-Cambrian mineralization exist in the Tertiary ores.

It has been pointed out that the sulfides are usually in smaller crystals in the Tertiary ores than in ores found in the pre-Cambrian rocks of the area. Given two host rocks of varying composition and texture, is it surprising that the sulfides are of different texture in each? "They (the ores) vary in texture, dependent upon local conditions in various Tertiary orebodies," writes Dr. Kerr, Columbia University[15] (personal communication, March, 1935).

The lack of rock alteration has been claimed to indicate a lack of Tertiary mineralization. One might ask how could the Tertiary solutions be expected to appreciably alter a series of metamorphosed pre-Cambrian rocks that had already reached the end phase of alteration long before Tertiary time? Little additional alteration could be expected; also, the lack of sericite and epidote, as pointed out, does not apply, as the invaded Homestake formation is not such as to produce these alteration products because it is not feldspathic nor highly calcareous. Some garnet is present.

Microscopic evidence is not conclusive as to what bearing, if any, the presence of introduced carbonates has on the age of the gold, as it has not been shown that gold and ankerite, for example, are related. It is true that where the values are low outside of the dike zone, ankerite is more abundant. This would suggest that pre-mineral ankerite may have been replaced in the ore zone. Paige shows a number of photomicrographs (*Bull.* 765) clearly indicating the replacement of carbonates by quartz and sulfides. It follows from his data that the carbonates in general preceded mineralization, and therefore their presence in the ores does not fit into any scale of later mineralization, whether pre-Cambrian or Tertiary.

Pre-Cambrian advocates have pointed out that the uniformity in the Homestake ores throughout considerable vertical range would point toward mineralization under conditions such as obtained in pre-Cambrian time, such as depth of cover, uniformity of pressure, temperatures, etc. It is held under the Tertiary theory that several thousand feet of cover existed and that sufficiently high temperatures and pressures to produce

the same result is evidenced by the presence of pyrrhotite, arsenopyrite and wolframite in the Tertiary ores.

Outlying occurrences of arsenopyrite, pyrrhotite and pyrite in quartz, which have been shown by mine development to be noncommercial, are frequent away from the rhyolite zone. It is held here that this points strongly to important Tertiary enrichment in the Homestake ore zone. This fact seems to be given little weight by the pre-Cambrian advocates, despite commercial failure of numerous attempts to mine what appears to be identical with Homestake ore in each direction from the main dike zone. Some examples are:

Surface	Direction from Main Dike Zone	Surface	Direction from Main Dike Zone
Bingham tunnel	Northwest	Rex mine	South
Columbia shaft	North	Pluma shaft	East
Gallagher prospect	West	City Creek	Northeast

Mine	Mine
Eighty ledge stope, 800-ft. level, northwest	Old Abe ledge, 2300-ft. level, east (and others)
No. 11 ledge, 800 and 900-ft. level, west	

The fact that the gold is associated with arsenopyrite and chlorite in the Homestake ore and the bearing of this on the age of gold introduction has been mentioned above. It may be added here that the pre-Cambrian advocates hold that the gold must have been introduced with these minerals. One Tertiary advocate holds that the sulfarsenides and sulfides are all of Tertiary age. He is certainly supported by the indisputable evidence of the presence of arsenopyrite, pyrrhotite and pyrite in the Tertiary ores.

Even should this suite of minerals in the pre-Cambrian Homestake formation be predominantly of pre-Cambrian age, there is no known criterion by which to judge the time elapsed after arsenopyrite deposition and the later addition of pyrrhotite, silica and gold. Figs. 3 to 9, inclusive, show clearly what several investigators, especially Kutz, regard as evidence favoring a later replacement of an older suite of minerals by silica, pyrrhotite and gold.

The fact of gold deposits of Tertiary age near and overlying the Homestake deposit is regarded by some advocates of the pre-Cambrian theory as having no particular bearing on the gold ores beneath in the pre-Cambrian rocks. Other pre-Cambrian advocates admit that the Tertiary solutions that must have transcended the Homestake ledges must have deposited some gold.

Advocates of the Tertiary theory, with this fact as a major premise, supported by microscopic work and broad field evidence, feel that most, if not all, of the gold was introduced during the latter period.

The presence of placer gold in the Cambrian conglomerate, if it could have been shown to be in important amount or even largely detrital gold,

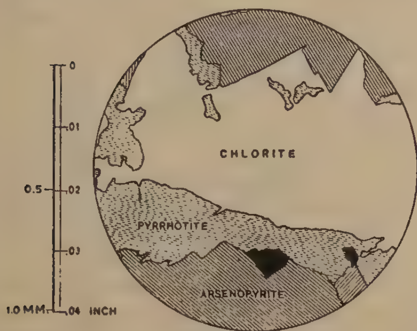

Fig. 3.

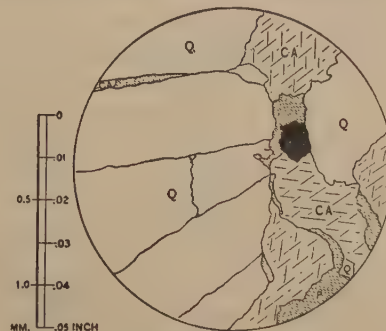

Fig. 4.

Fig. 3.—Pyrrhotite and gold replacing chlorite which has been replaced by arsenopyrite.

Fig. 4.—Showing carbonates filling fractures in quartz and replaced by pyrrhotite and gold.

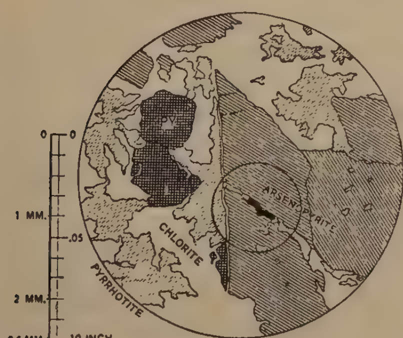

Fig. 5.

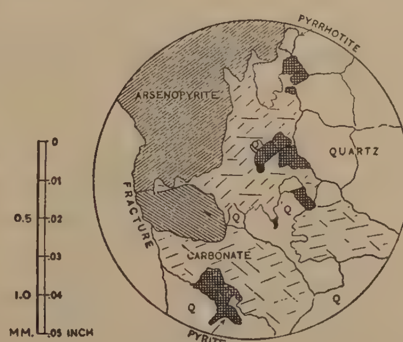

Fig. 6.

Fig. 5.—Pyrite, pyrrhotite and gold later than arsenopyrite, replacing chlorite.

Fig. 6.—Pyrrhotite, pyrite and gold replacing carbonate.
(*All shown by S. Paige, after W. J. Sharwood. From U.S.G.S. Bull.* 765.)

Table 3.—*Production from Conglomerate Ores*

Mine	Tons	Values (Estimated)	Oz.
Monitor	16,000	16,000 at 0.6 oz	9,600
Hidden Fortune (congl. only)	1,500	1,500 at 1.0 oz	1,500
Phoenix-Minerva	9,400	9,400 at 0.3 oz	2,820
		As placer from recent erosion of Monitor and Hidden Fortune deposits	11,100
		From small adjacent workings	1,000
			26,020

would have proved beyond the shadow of a doubt the presence of a pre-Cambrian lode. However, the evidence as given previously in this paper

does not seem to justify such a conclusion. In addition to this the total production from the conglomerate ores is estimated as shown in Table 3.

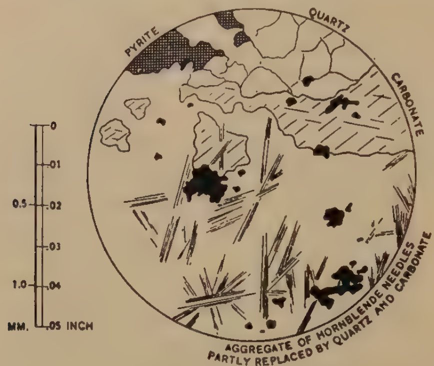

FIG. 7.

FIG. 8.

FIG. 7.—GOLD REPLACING QUARTZ, CARBONATE AND HORNEBLENDE.
FIG. 8.—TELLURIDE (BLACK) CONTAINING BISMUTH IN CONTACT WITH GOLD.
(*Both shown by S. Paige after W. J. Sharwood. From U.S.G.S. Bull.* 765.)

The Homestake deposit near the surface contained approximately 50,000 tons of 0.3-oz. ore per foot of depth, or 15,000 oz. If this gold deposit were in existence in pre-Cambrian time, the erosion of only 10 ft. of it would have produced 150,000 oz. of detrital gold.

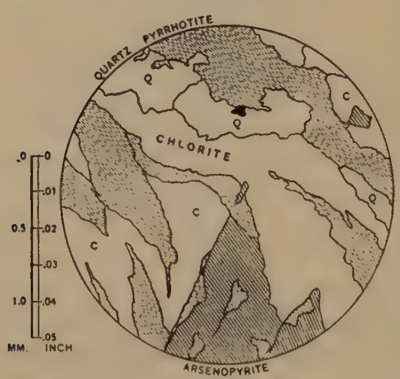

FIG. 9.—QUARTZ, PYRRHOTITE AND GOLD LATER THAN ARSENOPYRITE AND REPLACING CHLORITE. (*Shown by S. Paige after W. J. Sharwood. From U.S.G.S. Bull.* 765.)

With the conglomerate reaching a thickness of over 15 ft. within several hundred feet of the lode outcrop, certainly at least 100 ft. must have been eroded. This should have supplied 1,500,000 oz. of detrital gold, yet only 26,020 oz. are estimated to have been accounted for, or 1.73 per cent. And it is certain from Irving's evidence that much of this was replacement gold. Deducting the replacement gold, if we admit half was from this source, leaves 0.86 per cent *residual gold accounted for.*

This is excellent evidence that the Homestake gold deposit, as such, was not extant at the beginning of Cambrian time.

The fact of a generally higher silver-gold ratio in Tertiary ores seems to be well evidenced by analyses and ratios in bullion produced. However, it must be considered that the Homestake bullion is made up of ores taken each day throughout a vertical range of over 2000 ft. No comparable "sample" of Tertiary ores has ever been taken within three miles

of the Homestake. To contrast the ratios found in ores that occur in other mines of the district is not a fair application of evidence. It is a fact that the Tertiary ores *above* and within 2000 ft. of the Homestake lode are notably richer in gold as to silver than those more distant: example, Hidden Fortune (Grants mine). They were not in the class of refractory Tertiary ores found three miles or more distant. It is these distant refractory ores that pre-Cambrian advocates point to as being so radically different from the Homestake ore, obviously disregarding the fact that the early operators on Tertiary ores near the Homestake were able to make a satisfactory extraction by stamps and amalgamation. This could not be done even on the oxidized blue ores. Kutz[14] says:

> The presence of more silver in the Tertiary deposits indicates a shallower depth of mineralization and certainly is not a criteria for age. In connection with the zonal theory, it might also be mentioned that it is to be expected, if the Homestake is Tertiary, to find a large amount of silver in the zone surrounding the central gold-tungsten deposits, thus accounting for the difference in the gold-silver ratio. This difference in the gold-silver ratio also strongly suggests that the Homestake deposit fits into the zonal arrangement as outlined by Emmons.

Emmons, however, was not at that time satisfied to place the Homestake deposit in this arrangement, owing to the lack of tungsten in the ores.

A point has been raised relative to the presence of barite in the Tertiary ores and its absence in Homestake ores. Whether this has any bearing on the age of the gold seems doubtful. The sandy Cambrian dolomite probably contained barite as an original constituent. F. W. Clarke writes[16]:

> Barium sulphate has repeatedly been observed as a cement in sandstones . . . Clowes suggests that the barite was probably formed in situ, by double decomposition between barium carbonate and sulphates contained in percolating waters.

It has also been pointed out that the presence of tungsten in the Tertiary ore and its scarcity in the Homestake ore points to different periods of mineralization.

Fig. 2 shows the relative position of the tungsten ore and the Homestake ore. The tungsten-rich ores are separated from the Homestake outcrops by over 1000 ft. of intervening Tertiary oreshoots, which contained little or no tungsten. Also, that the tungsten mineralization is adjacent to a different zone of dikes. It may properly be ascribed to solutions different from those that deposited the Homestake gold, but of the same period.

The occurrence of tungsten 1000 ft. west of the Homestake lode was comparatively small and isolated. The total tonnage of gold-tungsten ore mined was under 150,000 tons up to exhaustion of the deposit. This, contrasted with over 50,000,000 tons of gold ore from the Homestake deposit, would seem to give ample ground for questioning its importance

in the scheme of paragenesis, except to conclude that a parallel system of conduits was traversed by solutions that contained some tungsten.

The "unusually" high percentage of K_2O in the Tertiary rhyolite has been mentioned as evidence that the rhyolite has not been a partner to the function of mineralization and rock alteration, especially in the matter, presumably, of sericitization of the invaded rock. W. J. Sharwood gives the analyses of Homestake rhyolite shown in Table 4.

TABLE 4.—*Analyses of Homestake Rhyolites* (*Sharwood*)

	1550-ft. Level	2000-ft. Level	From Deep Drilling
Silica	70.3	69.15	70.3
Alumina	14.3	14.4	14.4
Ferric oxide	1.7	0.65	Trace
Ferrous oxide			
Manganese oxide	Trace	Trace	
Magnesia		0.6	0.5
Lime	0.9	2.24	0.6
Soda	3.6	2.4	0.4
Potash	6.1	6.24	12.4
Water, 105° C	0.2	0.4	0.0
Ignition	1.3	2.64	1.0
Pyrite	1.8	2.13	1.6
Specific gravity	2.53	2.50	2.55

Analyses of eight rhyolites[16] from Maine to California, averaged, give: 74.46 per cent of silica, 13.43 per cent of alumina and 3.99 per cent of potash.

The Homestake rhyolite is lower in silica and higher in potash than the averages of others available.

If a surplus of potash argues that sericitization has not taken place, then why is it not closer to the point to assume that on account of an impoverishment of silica, silicification and mineralization has taken place?

Also, if it is to be expected, as has been held, that a high percentage of potash in the rhyolite is a criterion of nonexpulsion of mineralizers, the increase of 100 per cent in potash content noted in Sharwood's "deep" sample could well be interpreted as *indicating less mineralization* at that horizon than above. If this proves to be true, it would tend to indicate Tertiary mineralization.

It is true that the sulfide minerals are more abundant within the ore zone than without it. This, of course, applies to the Homestake formation, and not to the other sediments. The point is used in connection with the pre-Cambrian theory to stress the presence of an ore zone prior to Tertiary time, through which zone the Tertiary dikes have since been intruded.

It has been stated herein that the same suite of minerals occurs at a number of points without the main ore zone, but is lacking in commercial gold content. Workings lie outside of the main ore zone of which the arsenopyrite-rich rock does not contain gold proportionately with the same types from within the ore zone. The same holds for more distant occurrences of rock high in arsenopyrite.

The possibility of the rhyolite having intruded a pre-existing orebody has been attacked from many angles, as it has been realized by all students of this problem that herein might lie a definite criterion of relationships. E. S. Moore's observations have been given on this point, which would have definite bearing. He felt that the evidence was clear favoring post-rhyolite mineralization.

The contacts of the dikes with ore are chilled and often several inches to several feet of breccia are in evidence. The breccia, when lying between rhyolite and ore, contains fragments of both rhyolite and Homestake formation as well as fragments of other adjacent rocks, mostly slate and schist. Near the chilled margin of the rhyolite is usually a zone several inches wide that contains more pyrite than either the mass of rhyolite or the ore. Narrow veinlets of intrusive breccia containing a matrix of "mud" surrounding fragments of rhyolite and most of the other rocks of the pre-Cambrian series are found to crosscut both rhyolite and ore. This breccia is later than the contact breccias and has been observed crosscutting the former.

Fragments of pre-Cambrian quartz containing arsenopyrite in breccia were observed on the 400-ft. level, Caledonia orebody. It was evident that the arsenopyrite in that case existed before the brecciation. There were no known criteria by which it could be said that the gold, which the breccia contained, was not introduced during or after brecciation.

It must be admitted that the rhyolite dikes have the appearance of cutting the ore, in that ore is found on either side of the dikes. However, it must also be taken into account that the actual dislocation of ore minerals seems to be not in evidence, as the rhyolite contact is not sufficiently sharp or selvaged, but rather merges into the adjoining rocks, even if through a comparatively thin marginal zone.

That the Tertiary mineralizing solutions throughout the district arose adjacent to the rhyolite dikes (Fig. 10), of which there are numerous parallel occurrences, both east and west of the Homestake for several miles, is a physical field fact. It would indeed be a phenomenal coincidence if the Homestake deposit were the only exception and one in which this set of dikes (for the most part a single dike within the ore zone) bisected a pre-existing ore deposit *throughout a vertical range of over* 3000 *ft. and for nearly a similar average length.* If the pre-Cambrian theory is to be accepted, this coincidence must be admitted. And, in addition to this, it must be admitted that the dikes accidentally occluded the richest

portion of the main orebody, namely, the A and B ledges, which I and my associate, J. O. Hosted, first cut with a diamond drill on the 1850-ft. level in 1921, while exploring on the theory that a portion of an anticlinal fold of Homestake formation must exist within the dike zone at about this horizon.

Several hundred feet above this body of ore, a large fracture in the rhyolite contained mineralized breccia, oxidized sulfides, pyrolusite and psilomelane. Sharwood gives an analysis of psilomelane, which presumably came from this part of the mine: MnO_2, 66.66 per cent; MnO, 6.04; BaO, 15.40; H_2O, 6.44; WO_3, 1.00; Fe_2O_3, 1.90. (The BaO here is of interest in connection with placing the presence of barite in evidence supporting the theory of pre-Cambrian mineralization.)

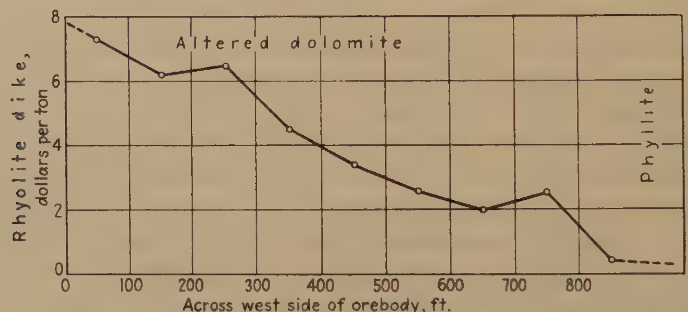

FIG. 10.—CURVE INDICATES THAT GOLD DEPOSITION DECREASED WITH DISTANCE FROM RHYOLITE ZONE[18].
Host rock occurs abundantly beyond limits indicated here on either side of dike.

A further study of the physical relationship of the dikes to gold distribution was engaged in and the following observations were published in part[17] in connection with a general theory of the relationship of dikes and ore, where the dikes and mineralizing solutions are related.

This article sets forth the fact that the rhyolite zone of the Homestake is consolidated into one main dike throughout the central portion of its length, but divides into a number of branches at either end, and that the ore occurs opposite and on either side of the "constricted" dike area (Fig. 11). Where the consolidation or restriction of the rhyolite is comparatively long, the length of mineralized Homestake formation is correspondingly greater, and vice versa. It is suggested that the physical features of structure that caused this constriction of the dike material had some direct influence on precipitation. Some figures are given (Table 5) as published in 1930 by permission of the management. The figures in Table 5 are from a wide vertical range and are not arranged in the order in which the variations occur.

It could, of course, be argued that ore lengths in pre-Cambrian time were determined by the same physical factors that determined the

disposition of rhyolite. If this is accepted, we must again admit that the rhyolite was intruded in such a manner as to bisect an existing orebody.

TABLE 5.—*Influence of Constriction on Precipitation*

Length of Constricted Area, Ft.	Length of Ore, Ft.	Ratio, Ore to Constricted Area
370	1280	3.46
140	480	3.43
1260	3050	2.42 (structural limitation).
290	920	3.28

If a central fracture or shear zone existed that determined first solution trend and later rhyolite trend, there should be some evidence of it in the structural pattern. Attempts have been made to show this, but so far as I am aware nothing of a definite nature has been found. It has been suggested that the height of pre-Cambrian surface west of the dike zone is 30 ft. greater than on the east and that this may represent the differential along a pre-Cambrian zone of movement. The difference of elevation is more logically accounted for by the regional slope of the pre-Cambrian surface from west to east.

A study of the structure maps of the mine, especially where definite horizons are cut by the dike zone, does not indicate any more displacement of structure than would normally be expected by the intrusion of a 150-ft. dike. (Fig. 11.)

That the dikes broke their way through folds and did not follow the pre-existing zones of shearing can be seen from the maps that Paige published in U.S.G.S. *Bull.* 765. The average strike of the rhyolite is N. 13 W.; average strike of ore, N. 13 W. average strike of footwall shear zone, N. 30 W. and average strike of axial planes of folding, N. 25 W. These averages are taken from the maps of each level to the 2300.

My associates and I attacked the problem in the first instance uninfluenced by either major theory, but with the thought that by having the proper key to the situation, the best service in ore finding and following could be given. As others became interested in the problem we were confronted by the dogma of high temperatures and deep-seated mineralization and it was suggested that our reasoning was unorthodox. After continued study and research, I concluded that if, in view of the above interpretation of evidence, the Homestake orebody could be admitted to be mainly of Tertiary age, it would be more in the interest of science to revise whatever of the ritual might be affected than to accept the easier task of following the previously printed word.

This paper should not be completed without some reference to the important work of Dr. Joseph P. Connolly[19], of the South Dakota School of Mines. Professor Connolly has dealt with the subject of mineraliza-

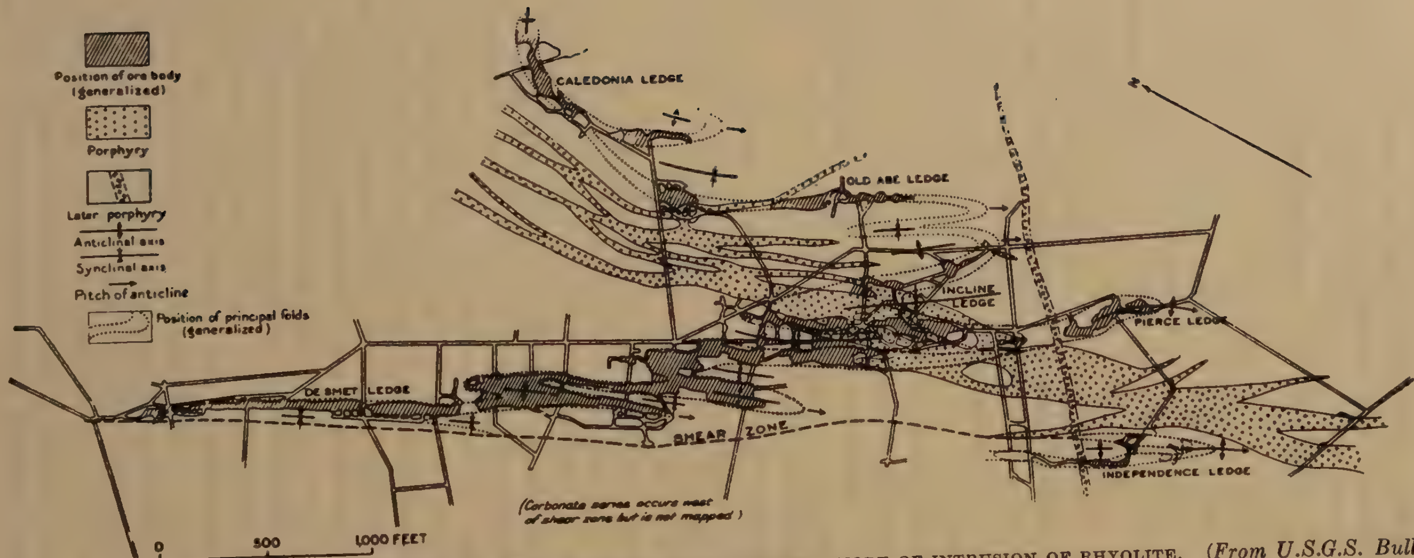

FIG. 11.—MAP OF 300-FT. LEVEL, HOMESTAKE MINE, SHOWING CHARACTERISTIC MODE OF INTRUSION OF RHYOLITE. (*From U.S.G.S. Bull.* 765.)

Note spur reaching into main zone of pre-Cambrian shearing. Also, lack of displacement of ore where folded host rock is cut by dike zone. Also, attention is called to lesser mineralization (not here indicated) as De Smet end of folds is approached, more distant from rhyolite.

tion in the Northern Black Hills as fully and completely as any investigator. His work points out the distribution and type of ore deposits that are definitely known to be of Tertiary age. He makes several impressive contrasts between the types of ore found in the Cambrian formation and the ore found in the Homestake mine. He concludes that the difference in associated minerals introduced is sufficient proof that the ores are of separate geologic periods.

His principal argument commences with the thesis that approximately 3500 ft. of cover existed over the pre-Cambrian rocks at Lead City at the time of Tertiary volcanic activity, and concludes from this that it is doubtful whether this was sufficient cover to permit the formation of high-temperature deposits below. It is doubtful whether he gives sufficient weight in this connection to the relatively massive sills that overlaid most of the deposits during their formation. These sills would naturally be at a comparatively high temperature and the solutions beneath would be impounded and subjected to conditions of cooling not at all comparable to those governing gold deposition in the very much cooler pre-Cambrian sediments below. The various points he raises in connection with differences in ore types are generally covered elsewhere in this paper.

In the matter of silver-gold ratio, Connolly gives some very striking figures from the Bald Mountain ores and, as is stated elsewhere herein, these ores occur 4 miles from the Homestake deposit, and it is a question in the minds of several whether the difference in silver-gold ratio is diagnostic evidence as to age of solutions. At the end of his bulletin is a series of photomicrographs, described as showing the following:

Plate 1. A. Homestake mine, 800 level, west ledge. Quartz with graphite inclusions compressed into minute folds. Thin section × 80.

B. Homestake mine, 1250 level. Veinlet of carbonate cutting quartz. Thin section × 80.

Plate 2. A. Homestake mine, 1250 level. Veinlets of carbonate cutting pyrrhotite but not cutting included magnetite. Thin section × 80.

B. Homestake mine, 800 level, west ledge. Carbonate with graphite inclusions compressed into minute folds.

Plate 3. A. Homestake mine, 1250 level. Pyrrhotite replacing cummingtonite. Thin section × 80.

B. Homestake mine, 1250 level. Pyrrhotite replacing cummingtonite between crystals and along cleavage. Thin section × 80.

Plate 4. A. Homestake mine, 1250 level. Pyrrhotite replacing cummingtonite. Thin section × 80.

B. Homestake mine, arsenopyrite replacing biotite and quartz. Thin section × 100.

Plate 5. A. Homestake mine. Chalcopyrite deposited on the boundaries between pyrite and pyrrhotite. A typical relationship in Homestake ore. Polished section × 80.

B. Homestake mine, 1250 level. Gold grains in arsenopyrite. Polished section × 344.

Plate 6. A. Homestake mine, 1250 level. Gold grains in arsenopyrite. Polished section × 80.

B. Part of same area as shown in A. Gold grains in arsenopyrite. Polished section × 344.

Plate 7. A. Homestake mine, 1250 level. Veinlets of pyrrhotite and gangue, quartz and chlorite cutting arsenopyrite. Polished section × 80.

B. Homestake mine, 1250 level. Pyrite, magnetite and gangue. Polished section × 80. (This section shows the magnetite largely surrounding crystal of pyrite. It is not certain from the photograph of the section whether the pyrite has replaced magnetite or not.)

The rest of the thin sections and polished surfaces are principally of Tertiary ores. They show a decided variation in texture for ore of this period from various localities. For example, in Plate 8, A; blue ore from the Portland mine, Bald Mountain area, is very much coarser than that shown from the Golden Reward mine in Plate 11, A. While the two have been labeled blue ore, it is to be noted that the Bald Mountain Co. has successfully treated the ore by the cyanide process and is still doing so, while the Golden Reward Co. found roasting necessary. In the ore treated by the Golden Reward Co. the very minute pyrite crystals seemingly could not be liberated by extremely fine grinding in sufficient proportion to permit economical attack by cyanide solutions. Connolly could not have meant it to be assumed that all the ores he classified as "blue" ore were "refractory" ores. Mention has been made elsewhere in this paper of a possible variation in texture due to the nature of host rock as applying not only to Tertiary ores but to the pre-Cambrian ore as well. In Plate 19 he shows two polished surfaces, one from the Galena district and one from Deadwood Lead and Zinc Co. In the former is shown löllingite and galena somewhat oxidized. In the latter he shows pyrite, arsenopyrite, galena and sphalerite. It is interesting to note that the presence of arsenic in both specimens checks with the observation of Kutz as well as that of Leaver and Woolf; i.e., arsenic in Tertiary ores.

I had the privilege of viewing many of these polished surfaces, and those showing arsenopyrite occluding grains of gold gave no hint that the gold might have been introduced after the arsenopyrite. So it is seen that we have evidence pointing to gold being contemporaneous with at least some of the arsenopyrite and definitely later in other cases in the same ore deposit. It may, however, be a reasonable speculation as to whether or not the gold may not have replaced arsenopyrite even though the section as cut shows an area of gold entirely and closely surrounded by arsenopyrite.

Connolly, after this study of the Black Hills ores, is of the opinion that the gold in the Homestake lode is principally pre-Cambrian in age. It seems to me that Connolly's conclusions incorporate many of the ideas that have been lumped as evidence of high-temperature mineralization when many of the gangue minerals named are undoubtedly older than

the mineralization and were formed during a previous period of metamorphism. The microscopic illustrations show definitely and are described as showing that the ore minerals are replacing the metamorphic gangue minerals.

Subtracting the metamorphic gangue minerals from the general set-up of mineralization, there is little dissimilarity in the final result in either of the gold ores.

Connolly also points out that "70 per cent of the Homestake gold is recovered by amalgamation. No primary Tertiary gold is so recovered." While this is a statement of fact as applying at the present time, it is a matter of history that 70 per cent of the Tertiary gold was recovered by amalgamation in and about the present Homestake in the early days. Connolly states that the gold is associated with various minerals, but particularly with arsenopyrite. This observation does not agree with the work of Paige, Kutz, Moore, and others who find that genetically a predominant portion of the gold associated with sulfides is associated with pyrrhotite, although often bordering crystals of arsenopyrite and filling fractures.

Connolly in his conclusions relative to the cement ores (conglomerate ores) makes no mention of the evidence Irving gives as to the replacement of the cementing material of the conglomerate by Tertiary solutions containing sulfides and gold. His last conclusion deals with the depth of cover, which has been commented on above.

In general, the work of Connolly has been painstaking and thorough, but his conclusions do not seem to square with all of his recorded observations, and in spite of the title, the work is obviously an effort to settle the question of mineralization in the Homestake, as the conclusions at the end testify.

Conclusions

Conclusions of the present paper are as follows:

1. The impure dolomitic strata comprising the Homestake formation was hydrothermally metamorphosed during the period of major folding with the formation of cummingtonite, garnet and chlorite, followed by widespread injection of vein quartz near the end of the folding period. This generation of quartz usually distinguished by being folded and relatively more strained.

2. Intrusion of the Harney (Algoman) granite, pegmatites and quartz with gold (as at Holy Terror mine) in the region adjacent to Harney Peak, the quartz generally less auriferous with increasing distance from the parent source.

The outlying zones of quartz and chlorite, as at the Homestake in the Northern Hills, being also nearly barren. Some occurrences of this generation of quartz contain massive pyrrhotite with poikilitic pyrite,

very low in gold. Others contain post-quartz arsenopyrite with marginal, undisturbed arrangement. (Microscopic evidence is very strong in support of later pyrrhotite, quartz and gold surrounding and crosscutting the arsenopyrite crystals.)

3. Long period of erosion and deposition of sediments, from Cambrian to early Tertiary, unconformably on the pre-Cambrian surface. The basal conglomerate covering wide areas, generally not auriferous, but containing a little detrital gold near the Homestake "reef."

4. Tertiary period of volcanism. Further uplifting of the Black Hills. Intrusion of rhyolite dikes into entire series, accompanied and/or followed by auriferous solutions. Superposition of quartz, arsenopyrite, pyrrhotite, gold and pyrite on the older quartz-chlorite-sulfide mineralization.

Replacement near rhyolite dikes of the calcareous matrix of the conglomerate with quartz, sulfides and gold. Replacement of the Cambrian dolomitic (and up to and including the Carboniferous) strata with silica, arsenopyrite, pyrrhotite, pyrite and gold with increasing amounts of silver. Increasing amounts of tellurium, silver, lead and zinc as distance is gained upward and laterally from the general center. Some very few of the many solution channelways at 1000 ft. or more distant from the main series giving rise to comparatively minor amounts of wolframite with the silica, sulfides and gold. The more distant deposits characterized by complete, rapid silicification and comparatively small sulfide crystals (refractory "blue" ore).

Also mineralization of fractures and shear zones in the Tertiary intrusives. (Example: Rattle Snake Jack and Gilt Edge Maid mines.)

5. The association of gold with arsenopyrite is shown by the thin section and polished surface records of three independent investigators to be largely due to the precipitation of pyrrhotite, quartz, pyrite and gold in and around arsenopyrite crystals as the precipitant. One weakness in this suggested sequence lies in the assumption of two ages of arsenopyrite. If it were present prior to Tertiary mineralization, there seems to be a decided lack of microscopic evidence of Tertiary arsenopyrite surrounding or cutting the older, as does quartz pyrrhotite and gold. That there definitely is Tertiary arsenopyrite in the Cambrian ores cannot be denied, and in proportions almost identical with that in the Homestake ore. This strongly argues the conclusion reached by Kutz, that all of the arsenopyrite was deposited in Tertiary time, and possibly even including the Bullion-Columbia deposit at Keystone, adjacent to, but in decided contrast to the Holy Terror pre-Cambrian quartz-gold veins.

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DISCUSSION

(*Edward Sampson presiding*)

S. Paige,* New York, N. Y. (written discussion).—The nature of the Homestake orebody is now well understood. In a word, geologists agree that this unique deposit is a selective replacement of definite horizons of pre-Cambrian carbonate beds. Many details of the stratigraphy, structure and mineralogy of these beds have been, for years, the subject of intensive study, on the surface and within the extensive underground workings of the mine.

But the age of the mineralization remains a matter of dispute. This fact alone should give one pause, more today than ever before, in any attempt to apply to the solution of the problem arguments not based on a dispassionate interpretation of field data, in the light of accumulated experience. Mr. Wright, I have no doubt, has attempted seriously to approach the problem in this spirit, but in his zeal he seems to have overstepped that considered weighing of the evidence that is the ideal of scientific research. In matters scientific, and this applies with great force today in the field of

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ore deposits, there is no obloquy in doubt. But Mr. Wright apparently is convinced that the Homestake orebody is essentially Tertiary in age, and he sets out to prove it.

At this time I beg the indulgence of my audience for a moment to refer to a matter that has caused me some amusement. Among those who have studied the Homestake orebody, some one, I do not recall the name, suggested that I had confused chalcopyrite with gold, the inference being that I could not, or did not distinguish between these minerals. Perhaps he was having difficulty with *his* determinations. I am sure I did not.

In what follows, I propose to briefly discuss the possibility of alternate interpretations of data presented by Mr. Wright; to re-present data he fails to bring forward; and to suggest what appears to me to be logical lines of attack on this elusive problem:

Placer Gold

On page 404, Mr. Wright estimates that at least 100 ft. of the Homestake lode must have been eroded prior to the deposition of the Upper Cambrian beds. He need not have been so modest in his estimates. It is more likely that thousands of feet were eroded from the pre-Cambrian rocks in the period of base-leveling that preceded the deposition of the Upper Cambrian. *But it does not follow that if there had been gold in the Homestake lode at that time, this gold would have remained near its origin.* Quite to the contrary, such gold would have been widely dispersed. Only very coarse gold would have remained behind, and even much of this gold also would slowly have moved outward during any long period of degradation.

I cannot, therefore, place much weight on the argument that the small amount of gold recovered in the Cambrian beds bordering the Homestake lode has much to do with the matter, except that it is a matter of record that experienced men reported the presence of gold that appeared to be of placer origin while at the same time recognizing that replacement deposits were present in the same beds. I refer to the reports of Devereaux and Irving, which already I have been at some pains to analyze. As with another great gold district, the Rand, the advocates of the placer origin (students on the ground—for most of the gold) are far from convinced, after years of study, that these deposits are epigenetic as is proposed by some serious workers, nor do I believe that Mr. Wright has proved that placers did not exist near the Homestake lode in Upper Cambrian time. If I am not mistaken, in general the gold of the Tertiary replacement deposits in Upper Cambrian beds occurs in very fine particles; for example: "The gold of the Cambrian ores except the conglomerate discussed further below is so fine-grained that no gold could be obtained in the pan—an almost universal characteristic of these ores."[20]

Why, in these deposits near the Homestake lode in the conglomerate at the base of the series was so much coarse gold found; in fact, nuggets? A difference that so far is unexplained except on the hypothesis that a placer deposit was enriched by Tertiary solutions.

Structural Considerations

On page 407 there appears the following statement: "The possibility of the rhyolite having intruded a pre-existing orebody has been attacked from many angles, as it has been realized by all students of this problem that herein might lie a definite criterion of relationships." I wish to say at once that much of the value of Mr. Wright's paper lies in a fair presentation of such data as he has been able to gather. Let us examine, therefore, carefully the statement of the third paragraph on page 407. Is there anything in this statement that precludes the hypothesis that the dikes are later than the

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[20] See W. B. Devereux, ref. 2.

ore? On the contrary, the statement is a striking suggestion that the dikes are later than the ore.

It is an unfortunate fact that in many instances age relationships as between minerals, or, in fact, between invading magmas and their metalized host, are elusive, because the argument can be and often is raised that "replacement" of deformed structures may confuse the picture. Thus it can and no doubt will be argued that ore deposition may have followed dike invasion, though the ore appears to be cut by the dikes. Or, if one cares to elaborate the argument—it may be argued that ore deposition just preceded dike invasion in Tertiary time and that the following dikes cut the ore. I would point out, however, that meticulous study of this relationship in the Homestake orebody should yield positive results. I am not impressed with Mr. Wright's argument that the dikes do not cut the ore. These dikes as portrayed on mine-level maps certainly appear to cut the ore. A casual inspection of the mine-level maps of the 200, 300, 400, 500, 600, 700, 800, 900 and 1000-ft. levels shown in my paper on the Geology of the Region around Lead, South Dakota, and its Bearing on the Homestake Ore-body[21] clearly suggests that the great orebody is unrelated to the position of the porphyry intrusions, and I have at some length pointed out that the orebody is fragmented and dislocated by intrusive dikes. On page 409 Mr. Wright says that the dikes broke their way through folds and did not follow the pre-existing zones of shearing, as can be seen from the maps that were published in *Bulletin* 765. I absolutely agree with this statement but my inference from it is just the opposite of that of Mr. Wright, as just pointed out. The dikes appear, in my opinion, to follow a fortuitous course, as many of the dikes do in the northern Black Hills. Is it not a fact that on the 300-ft. level there is a divergence of 22° or more between the course of the main dike body and the course of the main orebody from the Pierce ledge to the DeSmet cut?

There is another field of inquiry where intensive study should yield positive criteria. Is the Homestake orebody deformed? On page 45 et seq. of *Bulletin* 765, I presented evidence to show that this was true and illustrated this evidence with photomicrographs of polished specimens of ore and pointed out that in certain places it was hardly possible that replacement phenomena were simulating actual distortion of the ore. It is a fact that an important zone of shearing follows the west side of the main orebody, and whether this shear zone be called a fault or not is of slight importance. There is no question whatever that here the orebody was greatly deformed, within a very narrow zone, and, no doubt, displaced along this zone. On several levels, particularly the 900-ft., a mashed zone of slate in places 2 ft. wide may be observed in a position corresponding to the position of this shear zone. Within this 2 ft. of mashed slate intense folding and shearing has deformed the sulphides, pyrrhotite and pyrite, and also the gold. Masses of pyrite are sheared at their edges and drawn out in a finally divided state and mixed with slate. Gold has suffered similarly. The whole proves a movement under severe compression. No such forces as are indicated here have operated in Tertiary or later time in this region. These matters are not discussed by Mr. Wright with the thoroughness they deserve.

The occurrence of magnetite rather abundantly in several specimens of ores, *and clearly as a mineral introduced later than pyrite*, possibly affords further evidence of the pre-Cambrian age of the first mineralization. The magnetite occurs as fringing masses and crystals around pyrite grains filling re-entrant angles in groups of pyrite crystals and cutting or replacing crystals of pyrite. I believe magnetite has not been noted in the Tertiary ores overlying the Homestake lode, but it is a common mineral associated with the granitic invasion of the southern Hills both in pegmatite and in quartz

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[21] U. S. Geol. Survey *Bull.* 765 (1924).

veins. I regard its presence as confirmatory evidence of the pre-Cambrian age of the deposits.

Mr. Wright discusses the matter of tellurides, but I believe the following statements are even today undisputed. F. R. Smith has shown that tellurium is an abundant constituent of the Tertiary ores and suggests that sylvanite is probably the telluride present. Typical samples showed as high as 8.42 oz. of tellurium to the ton and the analysis of nine different samples show the following average percentages:

	Per Cent
Tellurium	59.97
Gold	7.64
Silver	32.39

At the Dacy mine, the percentages obtained approximate the composition of sylvanite:

Tellurium	61.20
Gold	36.27
Silver	0.53

In the Homestake ores tellurides are rare, though they have been noted. It is possible that the portion of the gold caught by cyaniding may be present partly in the form of tellurides. It is unnecessary to pursue the argument further. I may as well add, however, that I presented clear evidence of two periods of pyrite mineralization. The second well might represent the period of Tertiary enrichment. I believe today as I did 20 years ago that the evidence clearly supports the view that the first mineralization of the Homestake lode occurred in pre-Cambrian time and that some enrichment of the lode took place in Tertiary time.

J. J. Runner,* Iowa City, Iowa (written discussion).—Before presenting any new data that may have a bearing on the source of gold in the Homestake, it might be well to review briefly the points upon which there is a general agreement and upon which there is disagreement. The following seems to be quite generally accepted as the most probable sequence of events involved in the genesis of the Homestake formation and its ores: (1) deposition of the formation as cherty or sandy iron-magnesium carbonate-sediment with variable but small amounts of clay and calcium carbonate, (2) intricate folding and shearing of the formation and associated beds as great depths accompanied by dynamic metamorphism, (3) thermal metamorphism as a result of igneous intrusion followed by, (4) the introduction of sulphides, and finally, or at the same time, (5) gold. Such a sequence of events is one familiar to every student of gold deposits. In regions of such deposits he often finds the invading igneous rock, the aureole of contact and hydrothermal metamorphism and frequently a zonal distribution of ores partially surrounding the intrusive. He finds sulphides replacing carbonates and silicates and gold associated with the sulphides.

In many cases the geologist is not vitally concerned with the time elapsed between metamorphism and the introduction of sulphides nor between the latter and the deposition of gold. But in the case of the Homestake this time element is a very important consideration and is the point upon which Wright differs from Paige (ref. 21), Connolly (ref. 19) and McLaughlin[22] who have written concerning the genesis of Homestake ore. The differences of opinion have arisen out of the fact that instead of *one* there are *two* igneous intrusives in the region, both of which are quite certainly known to have been the source of mineralizing solutions bearing silica, sulphides and gold. I refer here to

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* Professor of Geology, State University of Iowa.

[22] D. H. McLaughlin: Ore Genesis and Structure (Chapter of the Homestake Enterprise). *Eng. and Min. Jnl.* (1931) **132**.

the pre-Cambrian pegmatitic granites centering about Harney Peak with the associated gold deposits of the Keystone region and the Tertiary rhyolites and quartz monzonites and associated gold deposits occurring as replacement deposits in Cambrian rocks in the northern Black Hills.

If Mr. Wright succeeds in establishing the Tertiary age of Homestake gold, he will have made a great contribution, not only to Homestake geology but to the science of ore deposits in general. It will cause us to review critically the evidence for the source of metals whose deposition has been *later* than the hydrothermal metamorphism and deposition of sulphides. We shall not so readily take for granted that a magma that has caused mineral changes has been responsible for the ore unless we can show that the time element demands it.

A comparison of the mineralogy of the Homestake orebody with that of ores formed during the Tertiary period has figured largely in the arguments for and against Tertiary mineralization of the Homestake. Too little attention, however, seems to have been given to mineralization of probable pre-Cambrian age in areas outside of the Lead region. It has been my privilege to study a number of these in some detail, and the data obtained will be reviewed briefly for whatever light it may throw upon the subject under discussion. The studies referred to, however, have been largely stratigraphic and petrologic and have had less to do with the subject of ore genesis. Altogether some 250 thin sections of mineralized rocks have been studied, a large proportion of which were assayed for gold and silver. No polished specimens were made to determine relationships among the sulphides, nor between them and gold. In a number, however, the former relationships were perfectly obvious with reflected light upon the thin section.

The similarity of certain pre-Cambrian rock formations outside the Lead region with those of that region was first suggested by Carpenter[23] in 1889. B. M. O'Harra,[24] Paige (ref. 5), Connolly[25] and McLaughlin (ref. 22) have reiterated the same belief for the rocks of the Rochford and Lead regions, and Paige and Runner[26] for other regions as well. While it is my belief that pre-Cambrian stratigraphy and low-grade mineralization in at least four areas in the Black Hills bear resemblances to those features in the Homestake, attention will be directed to but two of these areas. The best known and most intensively prospected areas are in the Keystone and Rochford regions.

Brief descriptions of the geology and gold deposits of the Keystone region have been published by Paige (ref. 5) and Connolly (ref. 25). More recently the geology has been studied and mapped in considerable detail by Hamilton.[27] Rocks called amphibolite by Paige and Connolly and regarded by them as originally of igneous origin are believed by Hamilton and the writer to be almost, if not entirely, the products of metamorphism of impure carbonate rocks. The area of gold mineralization of the Keystone region lies along the northeastern border of the area of outcrop of Harney Peak granite, from Iron Creek to Keystone, and extends beyond northeastward to the vicinity of Rockerville. The gold ores were largely in quartz veins but sufficient mineralization of the adjacent metacarbonate rocks to encourage much prospecting is more widespread. The original rocks contained sufficient clay and quartz so that

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[23] F. R. Carpenter: Ore Deposits of the Black Hills of Dakota. *Trans.* A.I.M.E. (1889) **17**, 570.

[24] B. M. O'Harra: The Gold-bearing Iron-quartz-tremolite Belt of the Black Hills. *Eng. and Min. Jnl.* (1916) **101.**

[25] J. Connolly and C. C. O'Harra: The Mineral Wealth of the Black Hills. S. D. School of Mines *Bull.* 16 (1929).

[26] J. J. Runner: The Pre-Cambrian Geology of the Nemo District, Black Hills, S. Dak. *Amer. Jnl. Sci.* (1934) **28**, 353.

[27] R. G. Hamilton: Unpublished doctor's thesis, State University of Iowa, 1935.

during deformation schistosity was developed in biotite, feldspar, quartz and carbonates, while garnet crystallized as porphyroblasts. Lenses and pods of granular quartz similar to the same, so common in the Homestake formation, are to be found in places. Thermal metamorphism caused the development of unoriented minerals including tourmaline, diopside, zoisite, fibrous amphibole, chlorite, brown and green mica, apatite, hornblende and titanite and occasionally white mica and microcline. The sulphides, arsenopyrite, pyrrhotite, pyrite and a little chalcopyrite, replace the silicates and carbonates.

In the Rochford region, the border of which lies some 12 miles northwest of the edge of the Harney Peak granite area, the stratigraphy of the pre-Cambrian sediments strongly resembles that in the Lead region. The Homestake formation possesses the same characteristic garnet, biotite, chlorite radiating fibrous amphibole and sugary quartz lenses and pods as it does at Lead. It is intricately folded and has been sufficiently mineralized in places to encourage some rather large-scale prospecting.

In the Rochford prospects, diopside, titanite and microcline appear to be lacking, and there is little zoisite, green hornblende, or white mica. Albite, orthoclase and ilmenite are common minerals. Rutile, tourmaline, apatite and graphite occur. Quartz, ferruginous carbonates, feldspars and biotite are early minerals followed by garnet, tourmaline, cummingtonite, green and brown biotite, quartz, bright green penninite and nearly colorless sheridanite. Ilmenite is later than cummingtonite, but its relation to sulphides is unknown. Small veins of nearly pure, brecciated albite, others of zoisite and quartz and still others of aragonite are to be found cutting the rock. Under the microscope, replacement veinlets of iron carbonate, pyrrhotite, rutile, apatite, chlorite and muscovite are seen to cut granulated aggregates of quartz and albite. This latter replacement has taken place in the granulated portions of a rock rich in albite, whether an albite vein or not is not clear.

The order of deposition of sulphides is, at least in places, arsenopyrite, pyrrhotite, pyrite. Penninite and sheridanite and occasionally biotite accompany late quartz, which in places appears to have been formed with arsenopyrite. There is good evidence that the chloritized portions of the formation contain the best prospects.

Paige (ref. 4), Connolly (ref. 19) and Gustafson[28] have written the most detailed descriptions of the petrology of the Homestake formation at Lead. Gustafson emphasized the paragenesis of silicates, Connolly the sequence of ore minerals, and Paige discussed both subjects.

The writer's studies have brought him into essential agreement with the data presented by these men, but not necessarily with all of their conclusions. Quartz and ferruginous carbonates were early minerals, the quartz occurring as lenses of sand or chert. Garnet, cummingtonite, biotite and chlorite were developed from the original materials by dynamic and thermal metamorphism. The sulphides, arsenopyrite, pyrrhotite, pyrite and a little chalcopyrite, replace the silicates and carbonates. Gold is found associated with the sulphides, chiefly with arsenopyrite. Zoisite, albite, orthoclase, apatite, tourmaline, muscovite, hornblende and titanium minerals are absent or rare. The complete and excellent published descriptions of the petrology of Homestake ores make further descriptions superfluous.

The persistence of sedimentary quartz and ferruginous carbonates, garnet, biotite, chlorite, fibrous amphibole, secondary quartz and carbonates, and the sulphides, arsenopyrite, pyrrhotite and pyrite in the mineralized rocks of the Keystone, Rochford and Lead regions make a genetic relationship of the three appear scarcely less than certain. The progressive falling off of the high-temperature minerals, diopside, zoisite, hornblende, feldspars, tourmaline, apatite, muscovite, and titanium minerals with

[28] J. K. Gustafson: Metamorphism and Hydrothermal Alteration of the Homestake Gold-bearing Formation. *Econ. Geol.* (1933) **28,** 123.

distance from the central area of Harney Peak granite appears consistent with the view that that magma was the source of solutions that developed the silicates and introduced the sulphides and gold into the Keystone and Rochford rocks and a portion, at least, of the same into the Homestake. The fact that the only important ore deposit in pre-Cambrian rocks in the Black Hills (the Homestake) is in the region of Tertiary mineralization, and that well recognized Tertiary ores of gold and tungsten are in most cases in very close proximity to the Tertiary acidic intrusives similar to the rhyolites that intersect the Homestake orebody, make the idea of additional mineralization of the Homestake by these Tertiary rocks an attractive one.

Irving (ref. 3) and Connolly (ref. 19) have written the most detailed descriptions of the character of Tertiary mineralization of the Paleozoic rocks of the northern Black Hills. A few data should be added to their excellent compilations in order to have a more complete picture of the situation.

Petrographic studies of mineralized quartz monzonite from the dump of the Cutting gold prospect on Deadwood Creek, ¾ mile above Central City, reveals the development of sericite, green mica, chlorite and calcite, and still later of sulphides as replacements of feldspars, pyroxene and amphibole of the original rock. Small veinlets containing chlorite, carbonate and sulphides are developed along shear planes. The green mica, chlorite and sulphide replacement suggests a similarity with late mineralization in the Homestake.

In a second prospect in the same rock on the north side of Sheeptail Gulch, approximately one mile northwest of the Cutting prospect, specimens from the dump show replacements of calcite, fluorite and sulphides. Among the latter was a notable occurrence of molybdenite. While the molybdenite is known in other types of mineral deposits, it is more common in pegmatites and hypothermal deposits than elsewhere.

In Two Bit Gulch, 2½ miles southeast of Deadwood, extensive silicification of the Cambrian beds has occurred at the contact with a Tertiary rhyolite sill. In the silicified zone are replacement crystals of hubnerite and orthoclase. Small irregular grains of tourmaline occur in the same, but whether they were detrital or secondary could not be determined.

These three instances prove nothing except the need of further studies on the petrology of the Tertiary intrusives where mineralization has occurred.

Considerable weight has been attached by the opponents of Tertiary mineralization to the chemical differences between Homestake and known Tertiary ores. Gold is known to be associated with the tungsten ores in the Cambrian in the old Hidden Fortune workings west of the Homestake and elsewhere. The deposition of wolframite is very much localized, whereas gold mineralization continues beyond the sharp borders of the tungsten replacements. So far as I know, no one has argued that the gold at these places when in association with wolframite is from a source different from that where no wolframite exists.

Another point raised by Wright, upon which there has been much comment and some difference of opinion, relates to the gold-bearing conglomerate at the base of the Cambrian sediments. Devereux (ref. 2) and Irving (ref. 3) state that they have identified in this rock rounded, smooth, slightly flattened grains of gold, which they interpret as certainty of detrital origin. According to Irving the gold was derived from the erosion of auriferous lodes in the pre-Cambrian rocks and was mechanically deposited in depressions along the ancient shore line. He says: " . . . this auriferous conglomerate . . . may be at once distinguished from the non gold-bearing portions of the basal conglomerate, as it is cemented by either oxide of iron in the weathered portions or by pyrite when it has not suffered alteration. The non auriferous conglomerate, on the other hand, has always a quartzitic, or in rare instances, a slightly calcareous matrix. The pyritic cement occurs in all the productive areas except one, and as all degrees of oxidation are present it can be assumed that the matrix of all of

the gold-bearing conglomerate was once pyrite." Irving believed that the ferrous sulphate formed by oxidation of pyrite had caused the dissolving of gold and "redeposition in thin films in the schists below. This has also produced an enrichment of the lowermost layers of conglomerate." Irving advances arguments for the introduction of gold with pyrite along "iron-stained fractures" now found in the roof of the stopes. This process has enriched the low and fairly uniform gold content of detrital origin. The introduction of pyrite was later than the deposition of conglomerate, since mineralization extends into fractures in the quartz pebbles. That this secondary pyritic mineralization was subsequent to rhyolitic intrusion is evident from the occurrence of rhyolites mineralized with pyrite, which cut the conglomerate in places.

To this argument for replacement origin of a part, at least, of the gold may be added the testimony of the tungsten ores occurring in the Cambrian sediments at Lead. These ores are normally pyritic, siliceous and gold-bearing. The writer has specimens of rhyolite from this region containing wolframite crystals and others of basal Cambrian conglomerate in which fine-grained wolframite ore has replaced the matrix surrounding the quartz pebbles. This tungsten mineral occurring in the conglomerate then is of Tertiary or later age, is associated with gold, and is of replacement, not detrital, origin.

In one of the above quotations Irving states that "the matrix of *all* the *gold-bearing* conglomerate was once pyrite." He cites assays to support his point that gold was introduced with this pyrite. If true, these data indicate a rather remarkable situation. They must mean that pyritic gold mineralization occurred *only* where detrital gold already had been deposited. This appears highly improbable. That the pyrite should be detrital seems equally unlikely. Could smooth, rounded grains and nuggets of gold be formed during replacement? Until someone can explain the origin of such forms as other than detrital, critics are likely to regard the form of these gold particles as evidence of the existence of a pre-Cambrian gold-bearing lode. A very careful reading of Irving's discussion of this point leaves one with the feeling that there was considerable doubt in his mind as to just what value should be placed upon each of the various modes of deposition of gold in the basal Cambrian. If he who visited the "cement" mines was uncertain, I do not see how his testimony can be regarded as a very strong argument for more than the existence of gold in pre-Cambrian rocks during the deposition of the basal Cambrian, certainly not for the existence of the present high-grade Homestake mineralization.

To sum up: The evidence cited relative to mineralization at Keystone, Rochford and Lead and the statements of Devereux and Irving, relative to the existence of detrital gold in the basal Cambrian conglomerate, indicate very strongly that there was pre-Cambrian mineralization in the Homestake. In my mind there is still considerable doubt of the relative value of pre-Cambrian and Tertiary mineralization, although I favor the former as being considerably more important. I do not believe that the data so far presented on either side of the case warrant a final decision. It would be far wiser to allow such arguments as are presented by Wright to stimulate one to greater effort to obtain the facts than to close the issue in favor of either hypothesis at this time.

L. C. Graton,* Cambridge, Mass.—My own familiarity with the Homestake is so limited that I can contribute nothing in the way of detail. But I should like to raise one or two general questions that may be pertinent to this controversy.

As Mr. Wright admitted, if one were to view the typical ore of the Homestake, which we know occurs in pre-Cambrian rock, the offhand conclusion would be that it is typical pre-Cambrian ore. That is to say, it is ore having the general charac-

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teristics, such as composition, texture and sequence (to say nothing of structure), that we are becoming accustomed to find in many regions of ancient rocks, where the evidence is more or less conclusive that the ore also is of great age. Although on some of those deposits mining has gone down to very substantial depths, virtually no systematic change has been revealed in the character or even the quantitative proportion of the minerals. To my mind, this implies that the modern erosion surface is a long way below the surface that existed when the now-disclosed portions of those ores were formed. For in ores deposited nearer to their then-existing surface, important changes in character are encountered at depths much less than the 5000 to 8000 ft. of the deepest mines, or than the present bottom of Homestake.

The Homestake mine is now about half as deep as the deepest. Throughout its explored vertical range it shows this same strong tendency toward constancy in nature of mineralization. This implies that the present erosion surface (which, near the mine, is close to the surface on which the Cambrian was laid down) was a long way below the surface that existed when the ore was deposited. And this, in turn, would mean either that the ore long antedated the Cambrian, or else, if the ore is Tertiary, that great erosion has since taken place from its initial outcrop. Inasmuch as the time available seems altogether inadequate for the latter of these alternatives, the presumption is strong that the ore is much older than the Cambrian.

The next question is: Do we know of any ores of the Homestake type in Tertiary rocks, or late Paleozoic rocks, or even in old Paleozoic rocks which have never been deeply enough buried to be substantially metamorphosed? Can we, in short, cite other examples truly of the Homestake type of mineralization that support the point of view that such ores can form reasonably near the surface? If such examples are not known, there seems to be another presumptive tally in favor of the view that the Homestake ores are deep-seated.

Finally, as between the admittedly Tertiary ores of the region and the Homestake ores, are the similarities cited by Mr. Wright sufficient to offset the great differences? The Homestake ores are, texturally and mineralogically, certainly of the high-intensity type; the Tertiary ores in the Paleozoic sediments with equal certainty are not. Yet the Homestake ores occur in relatively stable and chemically resistant schistose rocks, whereas the surely Tertiary ores occur in easily replaceable dolomite. If the kinds of country rock were reversed for the two groups of deposits, the differences in mineralogy and texture might conceivably be reconciled with the hypothesis that they are of the same age. But with the facts as they are, the fundamental differences between the two ores seem to me to be explainable only on the assumption that the Homestake ores were formed at very much greater depth than were the ores in the Paleozoic sediments. Therefore, they must be of very different age.

D. H. McLaughlin,* Cambridge, Mass. (written discussion).—An adequate treatment of the interesting problem of age of the Homestake mineralization would require consideration of far more evidence than has been presented in Mr. Wright's paper or could be conveniently included in brief comments upon it. The controversy, like many that plague geologists, is a difficult one to settle in a final way. On two important points—the hypothermal character of the ores and the later age of the dikes—the evidence seems to me to be absolutely compelling. Even so, I am quite willing to admit that a reasonable case can still be made for assigning all the gold deposits of the northern Black Hills to one rather long period of mineralization in the early Tertiary, though I do not find the arguments persuasive. No confirmation of predictions based on the hypothesis of Tertiary age, as advanced by Mr. Wright, has been obtained from developments in the mine in the past few years, as far as I

* Professor of Mining Engineering, Harvard University.

am aware, and I still favor the opinion, which I have previously expressed,[29] that the Homestake gold ores are of pre-Cambrian age. Critical consideration of the problem, however, must be postponed until data accumulated during the last five years by my associates on the staff of the Homestake Mining Company (Messrs. J. A. Noble, Chief Geologist, Clarence Kravig and J. O. Harder) can be fully presented.

L. B. Wright (written discussion).—Sidney Paige discusses further the character and distribution of the placer gold near the Homestake lode. He asks, "Why in these deposits near the Homestake lode in the conglomerate at the base of the series was so much coarse gold found, in fact, nuggets? A difference that so far is unexplained except on the hypothesis that a placer deposit was enriched by Tertiary solutions." The following facts will bear repetition:

The main production of coarse gold was derived from the Hidden Fortune (Grant's) workings, which are in part now accessible. The coarsest gold, from reliable information, came not from the conglomerate but from above the overlying bed of quartzite, several feet in thickness. The conglomerate horizon beneath was also very productive but was, as Irving states, mineralized by replacement of the original matrix. These ores are clearly described by Irving as characterized by veinlets of oxidized gold-bearing sulphides. The rich gold ore of the Grant workings graded into the hubnerite-gold replacement bodies in the same stratigraphic horizon.

We must also consider the production records from the district over a period of time. They indicate an average gold content of close to 7 dwt. per ton for all classes of ores. I have asked those who hold the view that most of the Homestake gold was in place before Tertiary time, "by what means was it possible for rich Tertiary solutions to passively transcend some 100,000,000 tons of pre-Cambrian Homestake formation, then effect replacements in overlying rocks of later age with a grade of ore similar to that below?"

Mr. Paige notes a divergence of 22° between the strike of the dike zone and that of the main orebody. This is correct, structurally, but the pattern of gold distribution within the host rock must be taken into account. When this is done, the major axis of gold distribution coincides more closely with that of the dike zone.

He also stresses the mashed slate zone with mashed sulphides and flattened gold. A very slight movement could accomplish this.

The presence of tellurides "widely spread" through Tertiary ores has not been generally demonstrated. Isolated occurrences are cited as bearing on the broad problem of ore genesis. I agree with Professor Connolly (ref. 19) that any attempts at correlation on this basis are futile.

Professor Runner presents evidence of a distinct change in hydrothermal effects in the pre-Cambrian rocks as distance is gained from the Harney Peak area in the central Hills toward the Homestake area in the northern Hills, some 30 miles. His observations tend to substantiate the possibility of a fading pre-Cambrian gold mineralization to an extent that may relate to the bodies of barren or low-grade Homestake formation containing quartz, chlorite, arsenopyrite, pyrrhotite and pyrite. These exist even within the range of mine workings on either side of the productive zone.

Professor Graton's remarks bring to the fore the general importance of this discussion and raise the question: To what extent can reliance be placed on the type of associated ore minerals in a deposit as a guide to geologic age? He concludes that the Homestake ore more nearly resembles in character and general geologic history deposits formed under pre-Cambrian conditions. This is logical, but does not pre-

[29] D. H. McLaughlin: The Homestake Enterprise—Ore Genesis. *Eng. and Min. Jnl.* (1931) **132**, 324–329; Ore Deposits of the Western United States (Lindgren Volume) 562–565 and 722–729, A.I.M.E., 1933.

clude the possibility of Tertiary enrichment of a nearly barren pre-Cambrian Homestake lode. With temperatures sufficiently high to permit the formation of equally profitable gold ores in post-Cambrian rocks above and adjacent (depth of cover notwithstanding), surely they were sufficiently high beneath.

Depths in the vertical range of 4000 ft. are not unusual in some post-Cambrian gold deposits where there is no pronounced change in host rock or structural control. The Carson Hill Gold Mining Corporation's Morgan-Melones mine at Melones, Calif., has been carried in Carboniferous rocks to a vertical depth of 4550 ft. with the records showing no decrease in ore grade. The vertical range of similar mineralization in the vicinity of Juneau, Alaska, is approximately 4000 ft. While these examples do not compare mineralogically with Homestake, they show that some post-Cambrian gold deposits have substantial depths. Also, the admittedly pre-Cambrian gold deposits of the Canadian shield display a wide variation of associated minerals in a region where factors controlling deposition are presumed to have been quite uniform, especially as to depth of cover.

Some students of the Homestake problem have suggested that the Tertiary rhyolite dikes followed an earlier Tertiary enrichment. Although the reverse has been held to obtain, further detailed study may show the dikes to have postdated the gold in Tertiary time. Some of the later phonolite dikes are known to definitely cut Tertiary ores in the Bald Mountain area 4 miles west of Homestake. Such a dike cuts the Homestake ore and rhyolite. In the Golden Reward workings, a rhyolite dike is obviously premineral with an ore shoot extending several hundred feet on either side. In nearly every case the Tertiary ores in the post-Cambrian rocks are adjacent to Tertiary intrusives in the entire northern Hills area. The genetic relationship must be intimate.

Professor McLaughlin's intimation that more recently accumulated data will be made available is interesting, especially in view of his remark that "no confirmation of predictions based on the hypothesis of Tertiary age, as advanced by Mr. Wright, has been obtained from developments in the mine, as far as I am aware, etc." I am not certain as to what particular prediction he refers.

It is to be hoped that full information on deep-level gold distribution may be made available, following which the accuracy of past predictions may be appraised and their bearing on this problem be finally considered.

Structure and Mineralization along the London Fault, Colorado*

By Quentin D. Singewald† and B. S. Butler,‡ Members A.I.M.E.

(New York Meeting, February, 1936)

Some of the broader relations between structure and ore deposition along the London fault, deduced from a thorough study of the geology of the eastern part of the Mosquito Range, should be of general interest to economic geologists. The first part of this paper is a generalized statement of the geology of the region in which the London fault is located; it may be supplemented by reference to several earlier publications by the writers and to the U. S. Geological Survey's *Professional Paper* 148, on the Leadville district. The second part deals with the structure of both the fault and the disturbed zone adjoining the fault. The third part discusses the salient features of the ore deposits, particularly emphasizing structural control. The two generalized maps are complementary to one another but contain too many data to show on a single small map.

The London fault is between Leadville and Alma, and the ore deposits along it are said to lie in the Alma district. Field work in this district, during six summers since 1927, has been a part of the cooperative work of the U. S. Geological Survey, the State of Colorado, and the Colorado Metal Mining Board. The writers have made use of work by R. D. Butler, J. W. Vanderwilt and R. E. Landon. G. F. Loughlin and C. H. Behre, Jr., who have worked on the west side of the range, have contributed valuable suggestions. Mining men of the district have aided in many ways.

General Geology

Pre-Cambrian Rocks.—The oldest rocks in the district are schists, correlated with the Idaho Springs formation of the Front Range. Quartz-mica schist predominates, but other types occur locally; almost everywhere the rocks show considerable lit-par-lit injection. Intrusive into the schists are two varieties of granite, only one of which is widespread. It is a fine-grained, porphyritic, gray granite correlated with the Silver Plume granite of the Front Range. Near the head of Buckskin Gulch a

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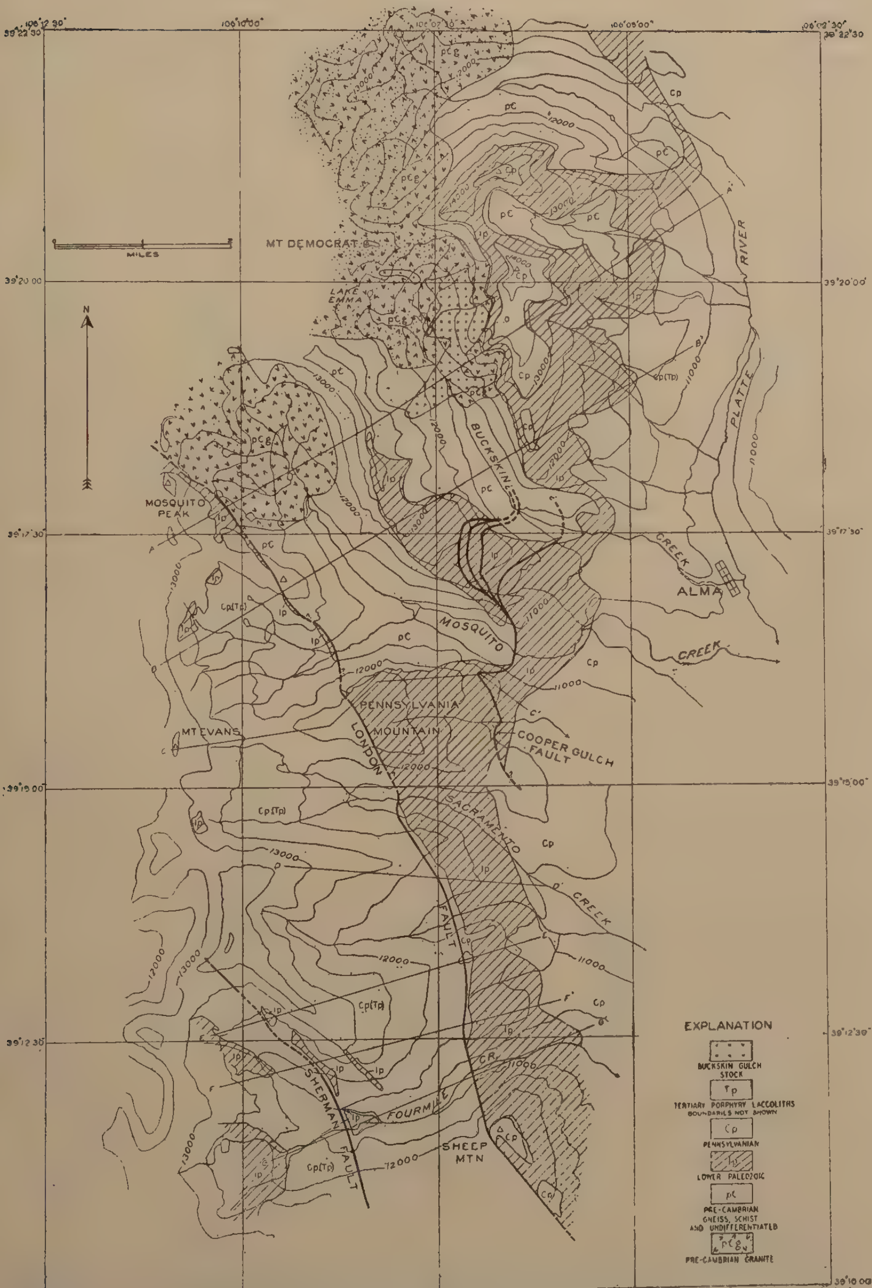

Fig. 1.—Generalized geologic map near London fault, Colorado.

granite gneiss having obscure to distinct foliation is mineralogically similar to and probably genetically related to the adjacent Silver Plume granite. Pegmatites are abundant in both the schists and the granites.

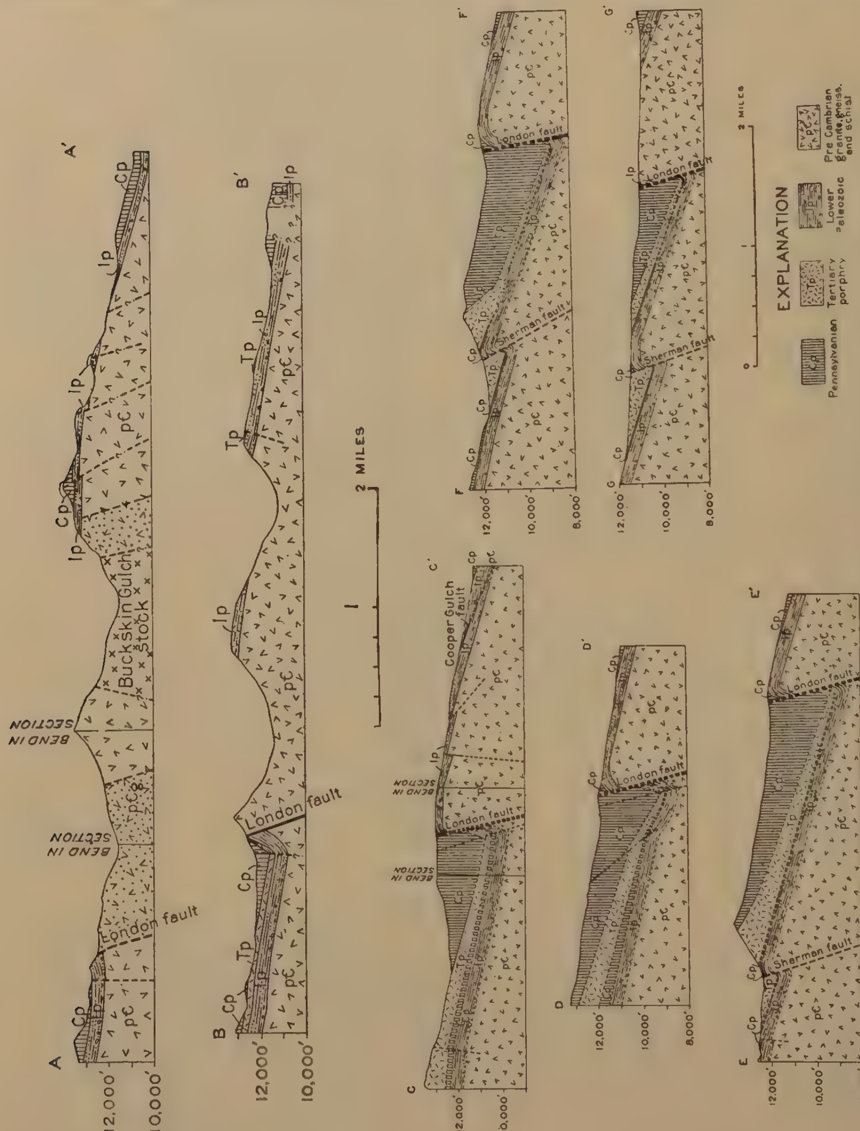

Fig. 2.—Generalized geologic sections across London fault, Colorado.

The areal distribution of the pre-Cambrian rocks is shown by the generalized map, Fig. 1, and the accompanying cross sections, Fig. 2. The pre-Cambrian rocks crop out only east of the London fault but are widespread in the northern part of the district. The distribution of the brittle granite and granite gneiss as compared with that of the more

plastic schist is noteworthy. Except for one major schist reentrant, granite predominates northwest and schist southeast of a broad arc that extends northeastward from London Mountain. As might be expected, this major contact served to localize a zone of shearing along which ascended much of the magma and most of the ore solutions during the later, "Laramide" orogeny.

Paleozoic Sedimentary Rocks.—In an economic classification the Paleozoic sedimentary rocks may be broadly subdivided into Pennsylvanian and pre-Pennsylvanian. The pre-Pennsylvanian rocks contain most of the valuable ore deposits in the Mosquito Range. In the Leadville monograph Emmons divided the pre-Pennsylvanian strata into five lithologic groups—in ascending order, the Lower or Sawatch quartzite, the Transition shales, the White limestone, the Parting quartzite, and the Blue limestone. These descriptive names have been retained by the local mining public, but the strata have since been further divided and reclassified as shown in Fig. 3.

The Pennsylvanian rocks along the London fault are part of the Weber (?) formation, composed of interbedded lenticular and commonly cross-bedded conglomerate, quartzite, arkose, grit, sandy shale, shale and limestone. Many of the beds are micaceous. Shale predominates in the lowest several hundred feet, and in a general way the abundance and coarseness of the conglomerates increase toward the top. The prevailing color is light, but much of the shale is black, most limestones are dark, some sandy beds are medium to dark, and some strata are red. The Weber (?) formation seems to vary in thickness from place to place, but must be at least 4000 ft. thick locally west of the London fault. The Weber (?) beds grade upward into the Maroon formation, of Pennsylvanian (?) and Permian age, of which the prevailing color is dull red. The Maroon crops out south and east of the area considered in this paper.

The distribution of Pennsylvanian and pre-Pennsylvanian rocks is shown by Fig. 1. The Weber (?) formation is widespread west of the London fault as far as the crest of the range, though older rocks crop out in local areas as indicated. East of the London fault the base of the main belt of Weber (?) rocks extends along a sinuous line northward from Fourmile Gulch. In addition, south of Sacramento Creek a narrow band of Weber (?) formation, locally interrupted where eroded, lies adjacent to the fault, owing to a reversal of dip.

Tertiary Igneous Rocks.—At about the beginning of Tertiary time the "Laramide" orogeny in the Mosquito Range was accompanied by intrusion of considerable quantities of magma. The resulting igneous rocks on the east side of the range may be classified into two main groups—the Buckskin Gulch stock, with its associated dikes, and the "intrusive porphyries." Both groups range in composition from diorite to granite, and doubtless all the rocks were derived from the same parent

Units Used in Field Mapping	Lithologic Zones	Thickness, Feet	Description	Units as Adopted by the Committee on Geologic Names		Age
Weber(?) formation			Interbedded quartzite, conglomerate, grit, arkose, shale, and occasional limestone, all micaceous. Some shale highly carbonaceous. Shale predominates near base, arenaceous beds in upper part.	Weber (?) formation		Pennsylvanian
				—Hiatus—		
"Blue limestone"	Upper limestone zone	0–160	Blue to black, mostly dense-textured, massive-bedded dolomitic limestone. Shatters easily. Weathers with pitted surfaces and breaks into blocky fragments. "Zebra-rock," chert, and limestone-breccia are common.	Leadville limestone		Mississippian
	Quartzite zone	0–8	Fine-grained to dense "cherty looking" white quartzite. Extremely lenticular.			
				—Hiatus—		
	Lower limestone zone	40–78	Fairly thin-bedded, mostly dense, white and blue dolomitic limestone. White beds weather cream-colored. Exposed surfaces generally smooth.	Dyer dolomite member	Chaffee formation	Upper Devonian
Parting quartzite		0–55	Cross-bedded and conglomeratic quartzite and sandy limestone. Quartz pebbles subangular. Locally, slightly shaly. Weathers light to dark brownish gray.	Parting quartzite member		
				—Hiatus—		
"White limestone"	Upper limestone zone	0–130	Thin-bedded white and medium blue, mostly "crystalline" dolomitic limestone. Weathers light gray, developing siliceous ribbing. Breaks to slabby fragments. Locally, slightly shaly at top.	Manitou limestone		Lower Ordovician
	Shaly zone	12–27	Interbedded dolomitic limestone, shaly limestone, and limy shale. Limestone weathers brown, shale green.		Sawatch quartzite	Upper Cambrian
	Lower limestone zone	15–30	Drab to brownish weathering limestone, dolomitic and somewhat sandy, with numerous limy shale partings.			
So-called Sawatch formation	Shaly zone	18–30	Thin-bedded, almost flaggy dolomitic limestone and shale. Upper limestones contain "red casts." Limestone weathers brown, shale green.	Peerless shale member		
	Upper limy zone	15–30	Brownish-weathering dolomiticlimestone with numerous shale partings.			
	Purple quartzite zone	2–15	Purple to nearly black quartzite. Slightly cross-bedded. Contains tiny angular quartz pebbles.			
				—Hiatus (?)—		
	Upper white quartzite zone	6–14	Fairly thick bedded, white fine-grained quartzite.			
	Lower limy zone	9–13	Thin-bedded series of quartzite, limy quartzite, sandy limestone, shale, and, rarely, limestone. Weather brownish and "sandy-looking."			
	Lower white quartzite zone	45–90	Fairly thick-bedded, white, fine-grained quartzite. A few beds have small quantities of carbonate cement. At base is a white quartzite conglomerate with pebbles less than 1" in diameter.			
Pre-Cambrian			Schist, injection gneiss, granite, and pegmatite.			Pre-Cambrian

FIG. 3.—GENERAL STRATIGRAPHIC COLUMN OF THE LONDON FAULT REGION, COLORADO.

reservoir. Petrographic descriptions of the stock[1] and of the porphyries[2] have been published elsewhere.

The stock is shown in Fig. 1. It was intruded contemporaneously with or later than the Cooper Gulch fault, which apparently influenced its location. The bulk of the porphyries were intruded before much faulting had taken place, and form sills in the sedimentary rocks and dikes in the pre-Cambrian. The sills are found at several stratigraphic horizons but are especially abundant in the Sawatch quartzite and in the basal part of the Weber (?) formation. Some of those in the Weber (?) are laccoliths as much as 1000 ft. thick; apparently at these places the magma could not force its way through the thick overlying shales. The latest of the porphyries—designated the "late White Porphyry"—was intruded after most of the faulting and forms dikes in the sedimentary rocks as well as in the pre-Cambrian. None of the porphyries are shown on Fig. 1, but some of the thickest sills are indicated on the cross sections in Fig. 2.

The stock and most of the porphyry dikes occur in a northeastward trending belt, not more than three miles wide, roughly coinciding with the main pre-Cambrian contact of granite and schist. Projected southwestward this belt also includes the Breece Hill and other stocks inferred at Leadville.

Structure

The regional dip of the stratified rocks in the Mosquito Range is eastward, away from the Sawatch uplift, but this dip is modified by folding and faulting. The most conspicuous folds are long and narrow and are associated with the major reverse faults. Each fault and fold may be considered to form a composite structure in which the early movement produced a fold, whereas the later movement produced a fault that was accompanied by additional drag of the beds along the fault. Folding of the more ordinary type is inconspicuous (Fig. 4) and is represented by variations in strike and dip that produce structural terraces and plunging anticlinal or synclinal noses. Economically, the most important of these are the terrace shown by the spacing of structure contours across Bross and Lincoln and the plunging anticline shown by the bend in structure contours of the same area.

The major faults on the east side of the Mosquito Range are shown in Figs. 1 and 4. The largest is the London fault, which, at least at London Mountain, is a reverse fault dipping steeply to the east. Associated with it is the London or Sheep Mountain anticline. The Cooper

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[1] Q. D. Singewald: Igneous History of the Buckskin Gulch Stock, Colorado. *Amer. Jnl. Sci.* (1932) [5] **24,** 52–57.

[2] Q. D. Singewald: Relations of Hydrothermal Alteration of Porphyries to Ore Deposition in the Alma District, Colorado. *Econ. Geol.* (1935) **30,** 518–539.

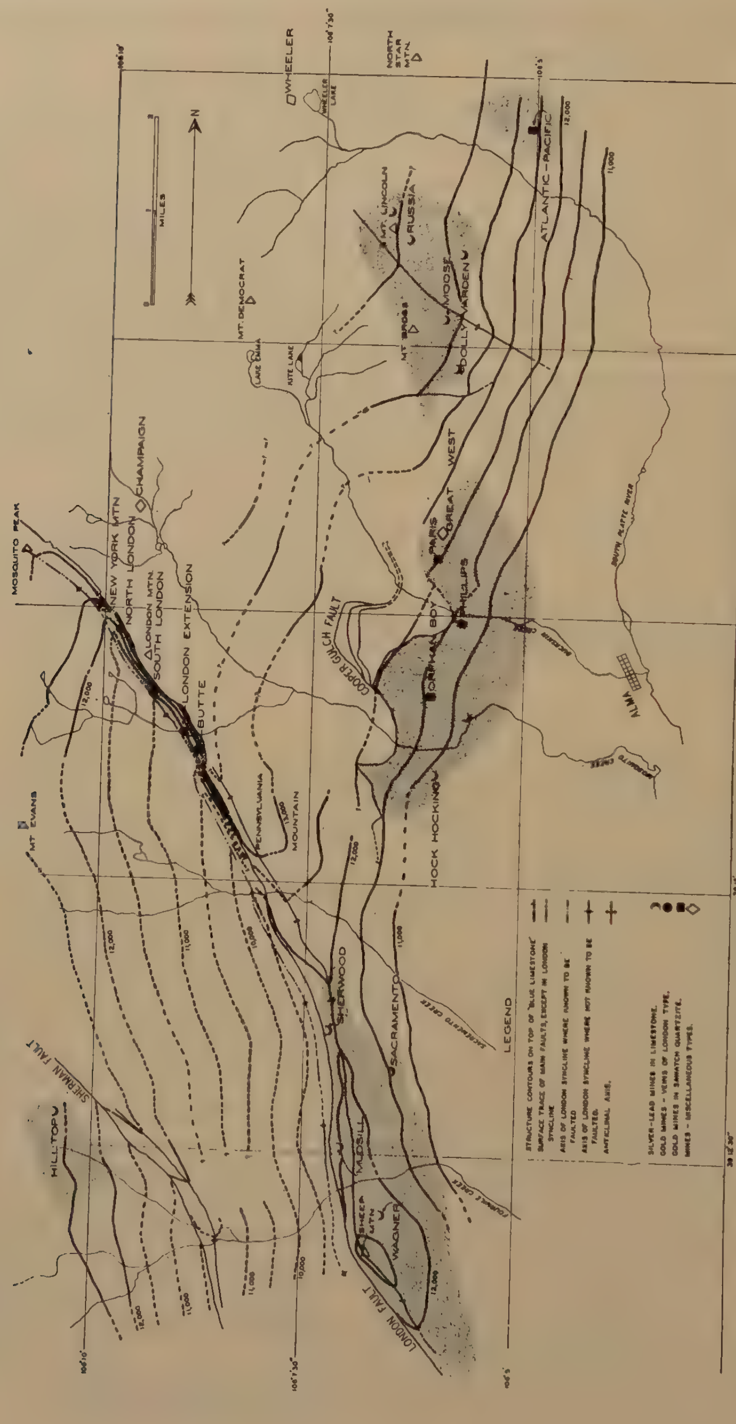

FIG. 4.—GENERALIZED STRUCTURE, NEAR LONDON FAULT, COLORADO. Areas of principal mineralization shown by groups of small dots.

Gulch fault, likewise reverse, dips much less steeply and has a smaller throw than the London fault; a small fold is associated with it. The Sherman fault, which is accompanied by a pronounced fold, is probably a reverse fault having an east dip, though it is nowhere exposed either at the surface or in mine workings. All available evidence indicates that most of the movement along the major faults took place prior to ore deposition, but that slight movements continued during and after ore deposition.

Minor faults, some auxiliary to major faults and others independent of them, are too abundant to be shown on small-scale maps, but a particularly large one south of Mount Bross is shown in Fig. 4. They are of economic importance, as virtually all the orebodies of the district are in or adjacent to minor faults.

Some additional explanation of the structure contours of Fig. 4 is necessary. The contours between the London fault and the London syncline on Pennsylvania Mountain are drawn a little west of their true position, in order to avoid the overlapping of the contours on beds east of the fault that would result from the east dip of the fault. Farther south no attempt has been made to sketch contours between the London fault and the syncline, because of the inadequacy of data. For the same reason, neither the surface nor the subsurface syncline west of the Sherman fault is shown. The intersection of the Cooper Gulch fault with the horizon that is contoured would be west of its surface trace as shown on the map.

London Fault

The London fault ranks next to the Mosquito fault as the second longest in the Mosquito Range. It has been traced southeastward from the Mosquito fault for more than 20 miles. It is associated with an asymmetric anticline of which the steep west limb, which nowhere is more than 2000 ft. wide, constitutes a disturbed belt in which the rocks have been sharply folded from their normal attitude and broken by innumerable auxiliary faults.

The fault, named from London Mountain, was first described in the Leadville monograph, in 1886, by Emmons. He recognized its large displacement and its close association with the folds at Sheep Mountain and elsewhere. It was not recognized as a reverse fault, however, until nearly 30 years later, when Moore and Patton had the opportunity to study it in the underground workings of the London mine.

Except in the main gulches, the position of the trace of the London fault can be determined rather closely at most places by means of exposures and prospect pits on each side. On account of a mantle of unconsolidated materials, however, outcrops of the fault are rare. One outcrop is on Sheep Mountain, between altitudes of 11,300 and 12,200 ft.

There the west wall is greatly shattered White porphyry, and the east wall alternately Parting quartzite and Dyer dolomite; both the Dyer and the Leadville are irregularly thinned by shearing and stretching in the fault. From west to east the shear zone consists of about 5 ft. of clay gouge containing crushed fragments of White porphyry, 50 ft. of coarse breccia composed of limestone, White porphyry, and clay gouge, and 75 ft. of irregularly silicified, broken limestone and limestone breccia. An outcrop of coarse, silicified breccia composed chiefly of Weber (?) quartzite forms a small but conspicuous butte at the top of the north side of Pennsylvania Mountain and extends 500 ft. vertically down the slope. The silicified breccia is flanked by and grades into unsilicified breccia, containing less quartzite, which is poorly exposed. A similar but less conspicuous outcrop of fault breccia occurs at the top of the south side of Pennsylvania Mountain. In places the width of the shear zone at Pennsylvania Mountain exceeds 100 ft. On London Mountain, just south of the point where the fault crosses the crest, fine-grained quartzite or chert crops out in the fault zone for a lateral distance of several hundred feet. At still another locality, on the south slope of a ridge that forms the divide between the two branches of Sacramento Creek, virtually no breccia exists, and so the exact position of the fault cannot be determined, in spite of excellent exposures of nearly vertical Weber (?) strata for more than 1000 ft. across the west limb of the fold. At all outcrops the fault has a steep dip, but neither the exact amount nor the direction can be determined.

Underground mine workings cut the London fault at several places. On London Mountain, Paleozoic sedimentary rocks and intruded porphyry sills to the west are separated from pre-Cambrian rocks to the east by a shear zone 40 to 100 ft. wide. Within the shear zone there are layers of highly crushed rock that alternate with thinner layers of nearly pure gouge. Locally, crushed pre-Cambrian rocks occur next to the footwall. Slickensides reveal innumerable planes of slipping. The dip of the fault, as revealed by observation and by projection in cross sections, ranges from 60° to 80° to the east. In the Berlin tunnel, which crosses the fault diagonally on the south side of Pennsylvania Mountain, the rocks are largely concealed by timber, yet it can be determined that the shear zone, 30 to 40 ft. wide, contains much heavy gouge, and that the Weber (?) rocks west of the fault and the pre-Cambrian rocks east of the fault are intensely shattered.

A significant structural feature is the regional southeastward slope of the sedimentary strata along the fault, due primarily to divergence between the trend of the fault and the strike of the beds. The slope is not uniform, however, as may be seen from the structure contours of Fig. 4. In the anticline east of the fault the pitch north of Pennsylvania Mountain is unknown, because the sedimentary strata there have been

entirely eroded. Across Pennsylvania Mountain the axis pitches rather steeply toward the north fork of Sacramento Creek, where it flattens abruptly. It continues with southeast pitch to a broad saddle at the south fork of Sacramento Creek, then rises very gently to a structural high on Sheep Mountain, beyond which it again pitches steeply southeast. In addition, there are local variations in pitch, some of which are suggested by bends in the structure contours.

The pitch of the axis of the syncline west of the fault, although complicated somewhat by faulting, also varies. It is moderate to the southeast from the crest of the range to New York Mountain, steep from that point to the crest of London Mountain, and then gentle as far as Pennsylvania Mountain, where again it steepens. In this interval, moreover, the curve in the contours midway between the portals of the South London and London Extension mines suggests a local flattening. Southeast of Pennsylvania Mountain altitudes at the top of the "Blue limestone" west of the fault are very uncertain. The most probable interpretation in the absence of subsurface data, however, is that the axis remains about level, at an altitude of about 9000 ft., as far south as Sheep Mountain. Surface dips in the Weber (?) formation suggest a structural saddle approximately beneath the Mudsill mine, with a very slight northwest pitch from that point to Sheep Mountain, but owing to undeterminable variations in thickness of porphyry sills near the base of the Weber (?), it is impossible to determine how closely dips on beds overlying the porphyries reflect the structure of the pre-Pennsylvanian strata. Neither the subsurface position of the fault nor contours between the west wall of the fault and the syncline have been sketched on Fig. 4.

In addition to variations in the pitch of the anticlinal and synclinal axes, there are even more decided irregularities in the pitch of the intersection of the fault with the top of the "Blue limestone" on each side. These irregularities are due chiefly to differences in the distances of the fault from the axes. For example, at the crest of London Mountain the fault is relatively far from the syncline, so that the dip, which becomes increasingly steep away from the axis, carries the top of the "Blue limestone" to a high altitude before it is cut off by the fault. Farther south the fault is closer to the syncline, and so the "Blue limestone" is cut off at a much lower altitude. Still farther south, toward South Mosquito Creek, the fault trends away from the synclinal axis, and the "Blue limestone" again rises higher, in spite of the southeast pitch of the syncline itself. Analogous but less pronounced features occur east of the fault.

The best data concerning displacement are obtained from the north side of Pennsylvania Mountain, where the total apparent throw—the vertical component of the distance between the anticlinal and synclinal

axes—on the top of the "Blue limestone" is approximately 3000 ft., of which about 1400 ft. is due to folding and 1600 ft. to actual displacement. Farther north the throw cannot be calculated, because the sedimentary strata have been entirely eroded east of the fault. South of Pennsylvania Mountain the Weber (?) strata west of the fault include no recognizable beds whose stratigraphic distance above the "Blue limestone" is known. By projecting the "Blue limestone" underground on the basis of surface dips, with allowance for thinning of porphyries near the base of the Weber (?), the total apparent throw at Fourmile Gulch may be calculated as about 3600 ft., though the limit of error may be about 1000 ft. Outcrops on cliffs on each side of the gulch show that at least 800 ft. of the apparent throw is due to folding and at least 1000 ft. to displacement. As the lower formations, where they disappear beneath the glacial drift in Fourmile Gulch, have dips nearly parallel to that of the fault, only half of the total apparent throw was assigned to displacement in constructing a geologic cross section (see Fig. 2). Between Fourmile Gulch and Pennsylvania Mountain, the calculations are still more conjectural but suggest that the total apparent throw remains about the same as at Fourmile Gulch, and so the actual displacement probably remains nearly constant also. Subsurface data, however, are necessary for precise information. The displacement along the London fault doubtless had, in addition to the vertical component, a horizontal component parallel to the fault, but data are not available to calculate its magnitude.

A tremendous number of auxiliary faults, which render the structure exceedingly complicated in detail, undoubtedly occur on each side of the London fault throughout its length. Exposures are not good enough, however, to observe more than a few of these auxiliary faults. Hence, an adequate study of them can be made only by detailed mapping of the extensive mine workings on the west side of the fault at London Mountain. There, the greater number of the auxiliary faults strike more or less parallel to the London fault but dip to the southwest; many, however, strike northwest and dip northeast, and many others are transverse. Some terminate against the London fault; others displace it. Movements on most of the auxiliary faults, even on many that displace the London fault, began contemporaneously with that on the London fault but were long continued. At several places on the surface moderately large transverse faults are suggested by apparent offset of the London fault for distances not exceeding 400 ft., yet none can be actually seen. Probably the most extensive belt of inferred transverse faulting occurs between the London Butte mine and the center of Pennsylvania Mountain.

Structural data in addition to those discussed in the text may be gleaned from the cross sections of Fig. 2, which include a few of the thicker porphyry sills and some minor faults not shown on Fig. 1. In

order to construct the sections from Pennsylvania Mountain southward, the London fault was assumed to dip steeply eastward, as at London Mountain, and the axial plane of the syncline was assumed to be parallel to the fault. Neither assumption, however, can be proved from surface data. Moreover, the dip of the Sherman fault and the position of the syncline west of it are not known.

Ore Deposits

Regional Distribution

Except for molybdenum, fully 99 per cent of the metal production in the Mosquito Range and over 90 per cent of that from the east side of the range has come from a belt of northeasterly trend, scarcely more than three miles wide, that extends from Leadville to North Star Mountain. This belt roughly coincides with but extends a little south from the belt of Tertiary dikes and the stock whose relation to the contact between pre-Cambrian granite and schist has already been mentioned. In spite of the general concentration, however, some valuable deposits have been found outside the belt.

In their order of importance, as measured by past production, the chief types of deposits on the east side of the range are gold veins of the London type, silver-lead deposits in limestone, gold deposits in the Sawatch quartzite, and miscellaneous types. Their general features have been described elsewhere[3]. As indicated by Fig. 4, the greater number of the deposits are grouped into six areas of principal mineralization. Gold deposits, if present, occur in the central part of an area; silver-lead deposits around the margin. Four of the mineralized areas are within the northeasterly belt, whereas two are south of it. The London Mountain and Sheep Mountain areas are closely associated with the London fault; the Mosquito Gulch-Buckskin Gulch area with the eastern branch of the Cooper Gulch fault; and the Bross-Lincoln area with a structural terrace that doubtless extends northward from the Cooper Gulch fault. Moreover, the area extending southwestward from the Hilltop mine lies south of a zone of faulting in Iowa Gulch, on the west side of the range. Only the North Star Mountain area is not associated with major structural features trending north to northwest.

London Mountain Area

The London Mountain area of mineralization contains the most valuable deposits of the entire district—the gold-quartz veins of the North London, South London, London Extension, London Butte, and other mines—which have produced a total of nearly $20,000,000 in gold,

[3] Q. D. Singewald and B. S. Butler: Suggestions for Prospecting in the Alma District, Colorado. *Proc.* Colorado Sci. Soc. (1933) **13**, No. 4, 103–110.

as well as considerable silver and lead and some copper. In addition, small silver-lead replacement deposits in limestone have been worked in New York Mountain, at the northern margin of the mineralized area. The southern margin has not yet been located.

The workings of the London mine cut mainly the upper part of the "Blue limestone," the basal part of the Weber (?) formation, and two interfingering porphyry sills—hereafter for convenience called the "porphyry zone"—ranging from 175 to 275 ft. in thickness, which occur 10 to 20 ft. above the base of the Weber (?) and contain included layers and lenses of the Weber (?) formation. These rocks are west of the London fault and are upturned against it. Their southwest dip becomes increasingly steep from the syncline to the fault. Depending on the distance between the fault and the syncline, the dip next to the fault ranges from 60° to slightly overturned. Innumerable auxiliary faults, already described, have broken the more brittle strata. Some idea of the structural detail may be obtained from geologic cross sections of the London mine, which have been published elsewhere[4].

The bulk of the ore has come from veins that lie stratigraphically between the top of the "Blue limestone" and the top of the porphyry zone, especially within the porphyry zone, but some has come from veins and replacement veins in the upper part of the "Blue limestone." The principal veins dip nearly parallel to the enclosing strata; they parallel the strata for some distance, then cut diagonally across to a higher stratigraphic position, and so on, as they dip away from the London fault. In places as many as four parallel veins are productive, but more commonly there are only one or two. Owing to the complexity of the structure, there is some uncertainty as to whether veins labeled "London," "McDonald," etc., were originally formed along continuous fissures or disconnected fissures at nearly the same stratigraphic positions. According to C. J. Moore, at one place in the northern part of the mine the London vein was stoped continuously for a distance of 1850 ft. horizontally, to a height ranging from 150 to 650 ft. Westward some oreshoots simply peter out, and others terminate against the contact, in places faulted, between the porphyry zone and the overlying impermeable shales. Eastward and upward the oreshoots may peter out, crop out at the surface, or reach the London fault. At the junction of a vein with the London fault the vein follows the sharply overturned strata into the footwall of the fault. Locally the oreshoots extend several feet up the footwall and slight mineralization may extend much farther. Here and there productive veins occur in fissures that strike northwest and dip either southwest, more steeply than the strata, or northeast. In at least two places veins that dip northeast in limestone join the London fault and continue laterally for 50 ft. or more in the footwall.

[4] Reference of footnote 3, plate in appendix.

The chief constituent of the veins is a slightly glassy variety of milky quartz, in microcrystalline to coarse grains. Pyrite, sphalerite, galena and chalcopyrite are subordinate constituents, and native gold is seen only in pockets of exceptionally rich ore. Insignificant amounts of calcite are widespread. In addition, the veins contain included masses, layers and filaments of partly replaced shale and porphyry. The paragenesis is: (1) wall-rock alteration, consisting of replacement by quartz and sericite, (2) coarse to medium-grained vein quartz, (3) brecciation, (4) fine-grained to microcrystalline quartz, (5) pyrite, (6) slight brecciation, (7) fine-grained quartz, (8) sphalerite, chalcopyrite, and galena, (9) gold, (10) slight brecciation, (11) fine-grained quartz and a little calcite, (12) slight brecciation, (13) calcite. All the quartz except that of stages 1 and 2 could be derived chiefly through granulation, solution and reprecipitation of earlier quartz.

The distribution of the orebodies close to the footwall side of the London fault shows that the fault has been a major factor in localizing the ore. Abundant evidence indicates that most of the movement along the London fault and also along most of the auxiliary faults took place before ore deposition, but that some movement, even on the London fault, continued during and after ore deposition. The relatively brittle rocks from the top of the porphyry zone to the pre-Cambrian rocks were very much fractured during the deformation. Prior to ore deposition this fracturing produced a permeable zone that extended several hundred feet from the fault and was bounded upward by impermeable Weber (?) shales and eastward by impermeable gouge in the London fault. The meeting of the two impermeable layers formed an inverted trough that pitches in general southeast, but locally northwest, owing to retreat of the fault from the syncline. Contrary to what might be expected, however, the distribution of the ore seems more closely related to flattening of the synclinal axis than to reversals in pitch of the inverted trough along the fault.

New York Mountain is at the northern margin of the mineralized area. Its production, which was small, came from silver-lead replacement deposits in the upper beds of the "Blue limestone." The original hypogene ore consisted chiefly of unreplaced limestone, iron-bearing dolomite, chalcedony, sphalerite, galena, pyrite, chalcopyrite, and a "gray copper" mineral; the sphalerite is lower in iron than that at London Mountain and contains no chalcopyrite blebs derived through ex-solution. The structure is essentially the same as at London Mountain, although where it is cut off by the fault the dip of the limestone is not so steep to the southwest. (See section *A-A′*, Fig. 2.) The region northwest of New York Mountain has been unproductive as far as the crest of the range, where some small mines suggest the possibility of another area of mineralization, in which the deposits, however, have been largely eroded.

Sheep Mountain Area

Associated with the doubly plunging anticline at Sheep Mountain is an area of mineralization extending from the saddle at the south fork of Sacramento Creek to a point nearly two miles southeast of Sheep Mountain. Eastward the area extends a mile beyond the London fault. Apparently after ore solutions once leaked through the fault gouge, they tended to migrate upward and outward into the hanging-wall country rock as far as fissures were available, and so the resulting deposits are neither as restricted in distribution nor as large as those at London Mountain. Moreover, the area is south of the main northeastward trending belt of mineralization in the Mosquito Range. All the productive mines are north of the high point on the anticlinal axis—in other words, on the gentle north limb rather than the steep south limb. The deposits are silver-lead replacement deposits in limestone, a type characteristic of the marginal mineralization in areas containing gold deposits.

The largest mine in the Sheep Mountain area is the Sacramento, of which the gross production has amounted to about $200,000. The orebody, close to the top of the "Blue limestone," was a replacement deposit extending up the dip from a minor fault that trends N. 30° ± E. Virtually no sulfide minerals were seen, but material on the dump indicates that iron-bearing dolomite, barite, and minor quartz were the chief hypogene gangue minerals in addition to unreplaced limestone.

The Mudsill mine is reported to have had a gross production of somewhat more than $60,000. Small replacement bodies of partly oxidized ore were mined immediately below a silicified (jasperoid) bed, 15 ft. below the top of the Dyer dolomite or Devonian part of the "Blue limestone." The orebodies dip with the strata, at an average of 50° SW., between the axis of the anticline and the London fault. No ore specimens could be found.

The production of the Sherwood mine amounted to nearly $50,000. The orebodies probably were replacement veins at or close to the top of the "Blue limestone," where the strata dip steeply to the southwest near the London fault. Iron-bearing dolomite and a little quartz were the gangue minerals in addition to unreplaced limestone. Sphalerite, galena, pyrite, a "gray copper" mineral, limonite and covellite were seen in specimens on the dump.

The Wagner mine produced decidedly less than the Sherwood. Small replacement orebodies were found in Dyer dolomite that dips 15° NE. The gangue consisted of unreplaced limestone, iron-bearing dolomite, and barite. Galena, light colored sphalerite, and some supergene minerals were seen in specimens on the dump.

The strata that crop out in the footwall across the fault from the Sheep Mountain area are stratigraphically well above the impermeable

shales in the lower part of the Weber (?) formation and therefore are unproductive throughout the Mosquito Range. Hence the absence of mineralization at the surface west of the fault does not preclude the presence of ore in more favorable strata at depth. The estimated depth to the top of the pre-Pennsylvanian formations is shown on the cross sections in Fig. 2. Whether these formations or porphyry sills near the base of the Weber (?) formation contain ore probably depends on the facility with which the ore solutions in this area could leak through the London fault gouge. If the solutions deposited most of their metal content on the footwall side of the fault, as at London Mountain, deposits on the hanging-wall side should be of a marginal type, like those in the Sheep Mountain area. On the other hand, even if no deposition took place west of the fault and the mineralization east of the fault utilized virtually the entire metal content of the solutions, the resulting deposits might be marginal in type, owing to the remoteness of the area from the main northeast belt of mineralization in the range. Hence the available data are not sufficient to answer the question, though the known presence of considerable gouge in the fault at Sheep Mountain and the large displacement on the fault suggest that the footwall area has some possibilities.

DISCUSSION

(*George M. Fowler presiding*)

R. D. Hoffman,* New York, N. Y.—The last time I was underground in the London district, in 1930, the limestones and the quartzites had flattened out to 20° and the fault had flattened with them, and the mineralization had pinched out. Has the structure changed since, or is that flattening continuing?

Q. D. Singewald.—Do you mean flattened toward the west side of the fault? Those formations do flatten out away from the fault toward the syncline axis. There are others that dip more steeply. The principal veins are the ones nearly parallel to the strata and they flatten along with the bedding, and where they flatten in two or three places the orebodies might peter out before the syncline.

* Mining Geologist.

Formation of the North-south Fractures of the Real del Monte Area, Pachuca Silver District, Mexico

By Edward Wisser,* Member A.I.M.E.

(New York Meeting, February, 1936; also Mexico City Meeting, November, 1936)

The Pachuca silver district, situated about 100 kilometers northeast of Mexico City (Fig. 1), covers roughly the southeastern half of the Sierra de Pachuca. The latter is a mountain range with northwest trend, and forms a part of the northeastern rim of the topographic basin known

Fig. 1.—Location of Pachuca district and other mining camps.

as the Valley of Mexico. The length of the Sierra proper is about 50 km., its greatest width about 9 km., and it rises to a maximum of 600 meters above the general level of the basin floor.

The Pachuca district consists of two contiguous subdistricts, the Pachuca area and the Real del Monte area. The center of the Real del

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Monte area lies some 6 km. east of the center of the Pachuca area (Fig. 2). The city of Pachuca, in the heart of the area of that name, lies in the Valley of Mexico, spreading up the southwest flank of the Sierra, while the town of Real del Monte (now Mineral del Monte) is on the northeast slope, close to the crest. The Sierra de Pachuca contains two other mining districts of some importance, the Capula and El Chico districts, which lie northwest of the Pachuca district, but they are, for the most part, outside the scope of this paper.

The Pachuca and Real del Monte areas really make one unit with regard to rock formations, intrusion and mineralization, but they differ in some respects, chiefly in their fracture patterns. This paper deals mainly with a phase of the tectonics of the Real del Monte area, but some consideration of the Pachuca district as a whole, of the entire Sierra de Pachuca, and even of the region surrounding that range, is needful for the light it brings to bear on the Real del Monte area.

General Geology of the Pachuca District

Extrusive Formations

The Sierra de Pachuca is made up chiefly of moderately dipping volcanic flows, breccias and tuffs. The basement on which these volcanics were extruded is not exposed in or close to the Sierra, so that the lowest members of the series are unknown. The stratigraphic section for 2000 m. above the lowest known horizon shows predominantly massive augite andesite, remarkably homogeneous in composition. Individual flows are commonly thick, but may be recognized, often by means of thin tuff beds at their tops and bottoms, or by layers of amygdules similarly placed. The rock is porphyritic, with glassy groundmass. These flows were extruded over an area considerably larger than that of the Sierra. Their source (or sources) is unknown. There is no proof that it lies within the Sierra de Pachuca.

In contrast to this wide distribution, the rocks younger than the andesitic series just described are limited to certain areas within the Sierra, one northwest, the other southeast of Pachuca (Fig. 2). Soda-rhyolite flows and rhyolite tuffs aggregating 250 m. in total thickness rest upon the andesitic series in these two areas. The sources of these flows are thought to be in the two areas in which they are found.

In the higher portions of the range the rhyolite extrusives rest on the older series with some erosional, but little or no angular, unconformity. Both series here dip gently, conforming with the nearly flat top of the range—a mountain range of accumulation, so far as its upper portions are concerned. But on the southwestern slope of the Sierra the rhyolite flows rest on the older extrusives with marked angular unconformity: they dip rather steeply towards the Valley of Mexico, while the gentle

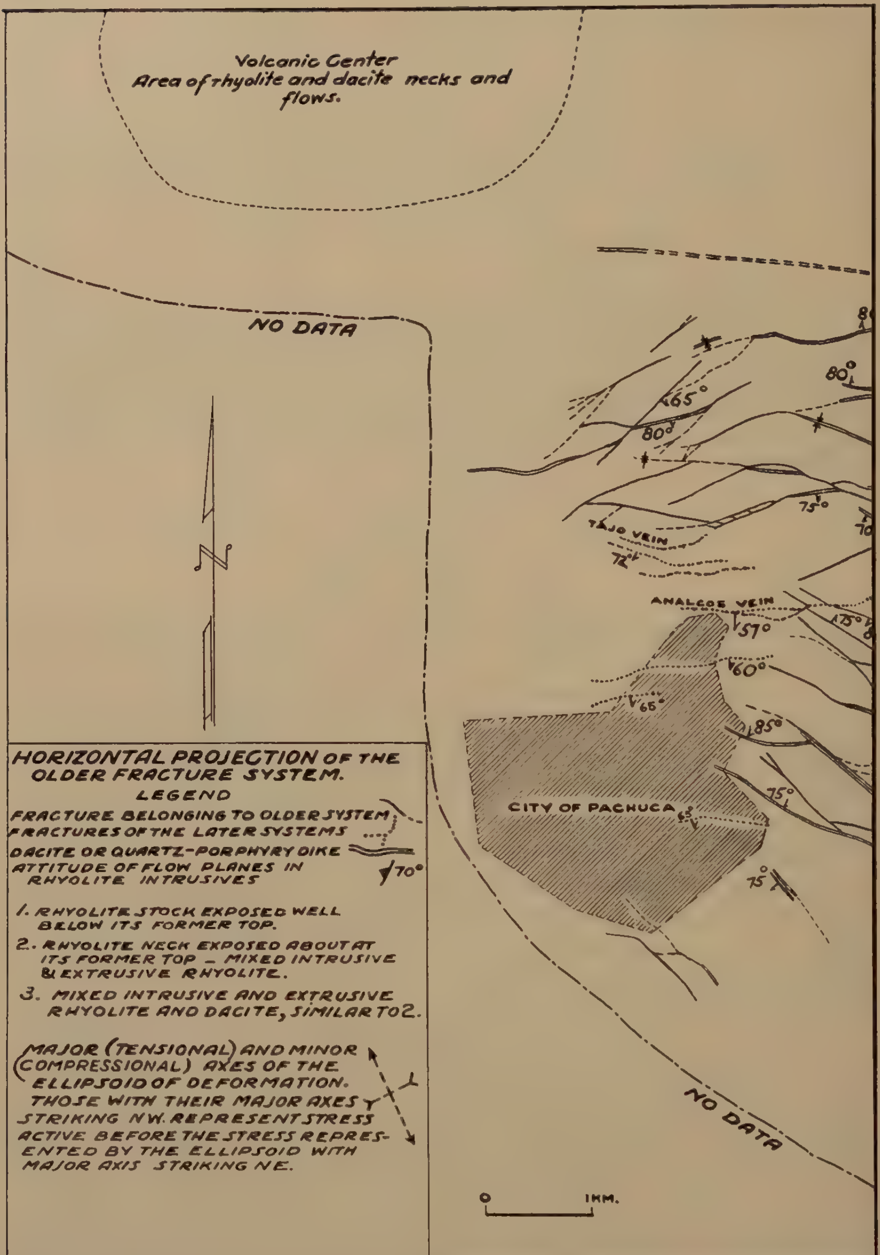


FIG. 2.—HORIZONTAL PROJECTION OF PRINCIPAL FRACTURES AND DIKES OF PACHUCA AREA.

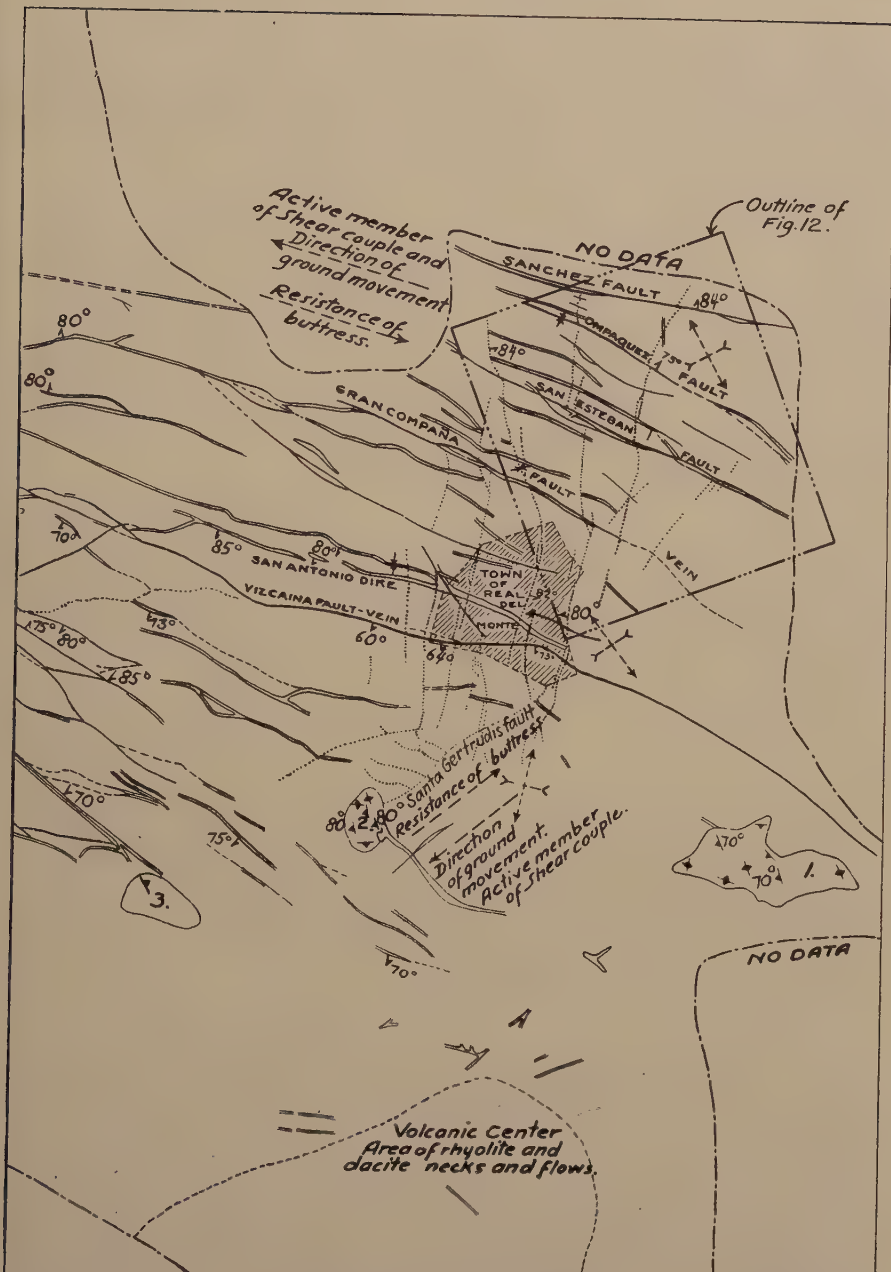


FIG. 2.—(*Continued.*)

dips of the older series show little change. This fact, with others beyond the scope of this paper, indicates that the Sierra was already an uplifted area, undergoing erosion, when the rhyolites were poured out on the crest of the uplifted area and down its sides. That this uplift was connected with an earlier intrusion that did not break through to the surface is suggested below.

Above the rhyolites, and resting on them with apparently perfect conformity, is a series of coarsely porphyritic dacite flows, grading in places to andesite and at the top horizons of coarse volcanic breccia. These rocks are confined to the rhyolite areas. Their maximum thickness is 300 m. They are thought to be the youngest rocks extruded except for decidedly later local flows of basalt.

All rocks up to and including the dacites are pre-mineral. The basalt is post-mineral.

Intrusive Rocks

Shallow-type intrusions, in the form of dikes and stocks, are extensively exposed in the Sierra de Pachuca and much less so in the lower country on the northeast side of that range. On the southwest side the Tertiary rocks are covered by the recent formations of the Valley of Mexico.

The oldest known intrusive type is coarsely porphyritic, quartz-rich dacite. It forms a number of dikes and occurs also as stocks, the largest of which, 7 km. northwest of Pachuca, measures 2 by 1 km. at the surface. The quartz-rich dacite intrusions took place prior to the outpouring of the rhyolite flows. The large mass northwest of Pachuca is accompanied by doming of the overlying flows, and by marginal thrusts. There is evidence, however badly obscured by later events, that an uplift of the Sierra de Pachuca area occurred at about the time of the quartz-rich dacite intrusions. It was upon this uplifted area that the rhyolite flows were poured out.

The quartz-rich dacite did not reach the surface, apparently, for no flows corresponding in age and appearance to these intrusions have been found. But the next intrusive surge, represented by the two areas of rhyolite (Fig. 2), broke through to the surface. Each area is believed to contain the channel or channels by which its rhyolite ascended. The evidence is not quite conclusive, for apparently in each area the present surface is very close to the horizon at which the rhyolite emerged from the crust and poured out over the then existing surface. Nevertheless, study of the areas points strongly to the conclusion that the rhyolite ascended in roughly cylindrical, nearly vertical channels. The two areas of rhyolite flows appear to be parts of old volcanoes.

There were two periods of rhyolite intrusion, of which the one just described was the first. No dikes of this first period have thus far been

identified; all dikes of which the age has been determined belong to the second period. But the fracture system along which nearly all the dikes of the district were intruded developed gradually, *pari passu* with the formation of the dikes. All types of earlier dikes are few in number compared to the great abundance of second-period rhyolite dikes, and a small number of first-period rhyolite dikes might well escape identification.

The intrusive next in age is normal dacite. This surge also broke through to the surface and formed the dacite flows, in the two rhyolitic volcanic centers. In the northwest rhyolite area volcanic necks of dacite are well exposed in one or two places; dacite flows were poured out over the rhyolite flows, and very coarse dacite breccia caps the dacite flows. Dacite in much smaller quantity reached the surface in the rhyolite area south and southwest of Real del Monte.

The dacite of this surge is found also as dikes, which cut through the rhyolite flows and at least the lower horizons of the dacite flows, their extrusive equivalent. The earlier, quartz-rich dacite dikes may be distinguished from the later dacite because they are capped by the rhyolite and dacite flows.

Closely following the intrusion and extrusion of normal dacite, a great number of quartz-porphyry and rhyolite dikes appeared, practically identical in appearance with the older rhyolite. This second rhyolitic surge occurred just prior to the mineralization, but overlapped the latter to a small extent. After the mineralization came minor and local intrusions of basalt.

Age of the Volcanics

The extrusives probably overlie, with marked angular unconformity, Upper or Middle Cretaceous sediments.[1] Such sediments, although not reached by mine workings, are found northeast, north and northwest of the Sierra. Imperfect fossil plant fragments in the rhyolite tuffs have been tentatively assigned to the upper Miocene. The volcanic series probably belongs in the middle Tertiary.

Structure and History of the Sierra

The structure of the Sierra—the folding and tilting on the one hand and the fracturing on the other—is closely connected with the series of intrusions and extrusions outlined above. The Sierra de Pachuca has been a locus of intrusion, diastrophism and mineralization. This locus of disturbance is not, however, an isolated one, but rather part of a major axis of disturbance, extending from Pachuca at least as far as El Doctor, Querétaro, 120 km. northwest of Pachuca (Fig. 1). A string of mining camps along this axis, including Cardonal, Zimapan, Jacala and El

[1] References are at the end of the paper.

Doctor, exhibit sharply folded and even overturned middle and upper Cretaceous sediments, the strike of the folds usually running north by west to northwest. Intrusives are present at Zimapan and Jacala, and their longer axes usually trend north by west. The general trend of the regional axis is N.35°W. The orientation of the sharp folds suggests a horizontal northeast-southwest compressive force.

The age of the deformation is post-Upper Cretaceous. W. T. Thayer[2] outlines a portion of the geologic history of this general region, in effect, as follows:

> The land mass emerged from the Gulf of Mexico at the end of the Cretaceous. Deformation followed along general north-south lines, with elevation to a height sufficient to be susceptible to active erosion (probably in the early Eocene). This erosion base-leveled the country and the resulting nearly flat plain is known as the Cordilleran peneplain. Then another uplift took place. This was accompanied in the beginning (probably in the Miocene) by orogenic movements that altered the entire structural trend to a general northwest-southeast direction. There also occurred at this time widespread volcanic activity, both extrusive and intrusive.

Thayer's age for the diastrophism, intrusion and extrusion, the Miocene, corresponds to the probable age of the rhyolite extrusions of the Sierra de Pachuca. At Zimapan, Jacala and other districts northwest of Pachuca the Miocene diastrophism was marked by compressive stress acting in a direction about N.55°E.-S.55°W. It would not be surprising, then, if the results of such a stress were found in the Pachuca district, since most of the exposed rocks there antedate or are contemporaneous with the diastrophism.

Considering the concentration of shallow intrusives in the Sierra, the volcanic centers there, and the large intrusives of more deep-seated type that have emplaced themselves in the Cretaceous sediments northwest of Pachuca along an axis of disturbance of which the southeastern prolongation passes through the Pachuca district, it is very probable that a large intrusive mass lies beneath the Sierra de Pachuca.

The earliest and still dominant fracture system in the Sierra de Pachuca strikes, on the average, between N.75°W. and east-west. The more northwesterly strike in the Pachuca district, Fig. 2, is a local variation. This system is coextensive with the locus of most intense intrusion, the Sierra de Pachuca; it kept on developing while younger and younger dike types were intruded. The system has, therefore, probably a genetic relation to the supposed major buried intrusive mass.

Hans and Ernst Cloos, Robert Balk and others have made extensive studies of granitic intrusives. The most comprehensive exposition of the ideas of the Cloos school I could obtain is supplied by three general papers by Robert Balk[3,4,5]. These workers show the following sets of fractures formed during the emplacement and solidification of granite massives, and shortly after the solidification.

1. *Cross Joints.*—Wherever an intrusion develops flow lines . . . tension joints can be recognized, running perpendicularly to the direction of maximum lengthening . . . they have been termed *cross joints* . . . They must have developed so soon after the mass crystallized that they reproduce exactly the same distribution of stresses as the flow lines. Strike and dip of cross joints vary harmoniously with any variation in trend and pitch of flow lines. [The strike is perpendicular to the trend of the flow lines, the dip complementary to their pitch.] In massifs where flow lines form an arch, the cross joints are arranged into a fan structure . . . the individual planes dipping into the intrusive along the margins, and stand more or less vertically near the apex of the flow-line arch. . . .

In massifs that do not develop flow lines, cross joints in the strict sense cannot be recognized. It is possible, however, to designate a system of *regional tension joints*, if a direction of maximum crystal compression is recognizable. This is usually easier than may seem possible, because the tension joints tend to carry early dikes. . . . [4]

2. *Flat-lying Normal ("Antithetic") Faults.*—A mass that has begun to freeze near the top, but continues to be pushed up by a liquid residue below, may be expected to gain the necessary space increase by faulting along the cross joints . . . While many cross joints have subsequently served as faults, experience shows that auxiliary faults are formed that differ from . . . cross joints in their directions . . .

As first found in granite massifs in Silesia, these faults dip at moderate angles in random directions. Displacements along them prove to be normal faults in every instance, and the *trend of the striae on them coincides with the trend of the flow lines in the granite.* Since the movements on these planes had obviously helped to expand the granite in the same direction as that in which the mass had flowed, the term 'planes of stretching' [German: *Streckflächen*] was used alluding to *Streckung* (stretching), the German equivalent for flow lines.[4]

3. *Marginal Fissures.*—Along contacts of steep-walled intrusions, a group of fractures has been found . . . The fractures dip at intermediate angles into the intrusive, may extend a small distance into the wall rocks, and appear within a mile or so from the contacts . . . Marginal fissures commonly carry aplites, pegmatites or quartz veins, and develop as early as cross joints or flat-lying normal faults . . . The vertical zone, parallel to the contact, in which the fissures occur, represents a zone of maximum strain (due to the upward motion of the magma mass, on the one hand, and the retarding effect of the stationary contact wall, on the other). The individual fissures are tension joints with reference to the lines of maximum . . . lengthening.[4]

The generalizations developed, mainly by Hans Cloos, concerning antithetic and synthetic movements, furnish valuable criteria for the determination of the stresses operating to produce certain types of deformation. "Antithetic movements are smaller movements that are associated with larger movements, but have the opposite tendency.[4]" Fig. 3 contrasts antithetic and synthetic movements. With antithetic faults (dip-faulting) some horizontal distension or compression of the crust is needed, to permit or cause the rotation essential to this type of faulting; with synthetic faults, some vertical force is needed to produce the results seen. As Balk says, in effect, the prevalence of antithetic faults in domes or troughs implies a certain amount of ability of the crust to yield horizontally (Fig. 3*a*); the prevalence of synthetic faults

implies an uplifting force, or the ability of some of the blocks to slip unhindered downwards (Fig. 3*b*).

Fig. 4 is a cross section of the Sierra de Pachuca N.75°W. fracture system, constructed about at right angles to the strike of the fractures

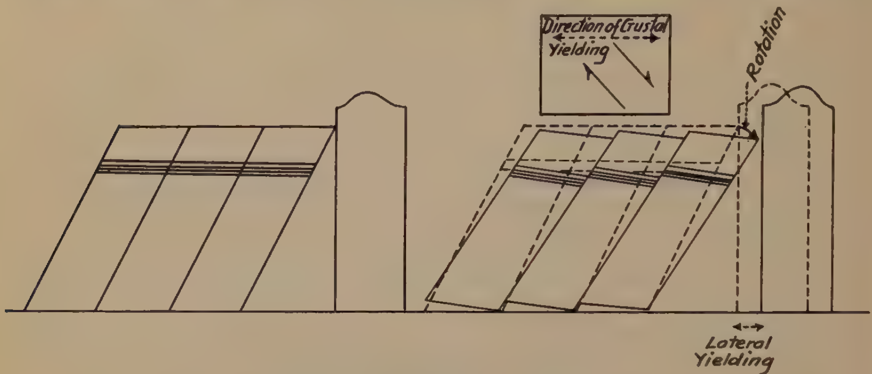


FIG. 3*a*.—ANTITHETIC FAULTING.

Row of books on shelf. Backs of books are not rectangles, but parallelograms. Titles form a continuous horizontal line. Row of books rests against book-end. When book-end is moved to right, books rotate because of gravity, and come to rest again, right-hand book against book-end. Adjoining sides of any two books form two walls of antithetic fault. Displacement on fault tends to counteract effect of rotation upon line of titles (which may be compared to a sedimentary bed). The vertical movement of the books results solely from the rotation. Movement is due to release of lateral confinement and to gravity.

If books stood backs up on table—i.e., if figures were plans instead of sections—a clockwise shearing stress (shown in box, right-hand figure) would produce effect shown. Direction of yielding for plan is the same; it corresponds to direction of yielding when the plan is turned into a section.

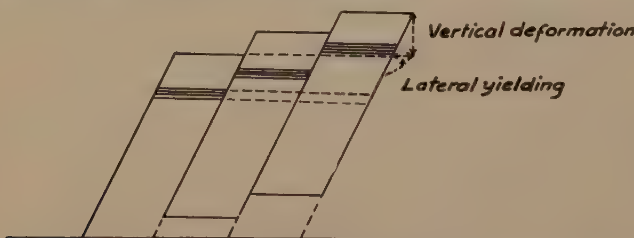


FIG. 3*b*.—SYNTHETIC FAULTING.

If same set of books is dislocated without rotation, some of them must either have been shoved upwards or allowed to drop freely. Movement was strictly along fault planes, instead of mainly almost at right angles to them, as in the antithetic faulting. Same thing applies if Fig. 3*b* is considered a plan. Displacement must have been by shoving along fault planes. Some lateral yielding is required here, as in antithetic faulting.

and the strike of the gentle folds in the extrusive formations. An area between Pachuca and Real del Monte, essentially unaffected by the later disturbances to be described, was chosen.

The section shows that the fractures dip steeply. The synthetic nature of the faulting is obvious. Instead of tending to counteract the direction of deformation of the extrusives (anticlines or synclines), as

antithetic faults would, these faults accentuate by their displacements the anticlines and synclines. A vertical force is needed to produce such faults. The presence of vertical force at the time of deformation is suggested by the steeply dipping reverse faults, without using the conception of synthetic faulting; but it is to be noted that if these faults had dipped north instead of south, showing the same displacements, they would have been normal faults, geometrically considered, but still synthetic according to the broad and convincing generalization of Cloos. Further evidence for vertical force is found in the folds themselves. Such gentle folds in brittle rocks could scarcely have been formed by forces acting otherwise than nearly vertically.

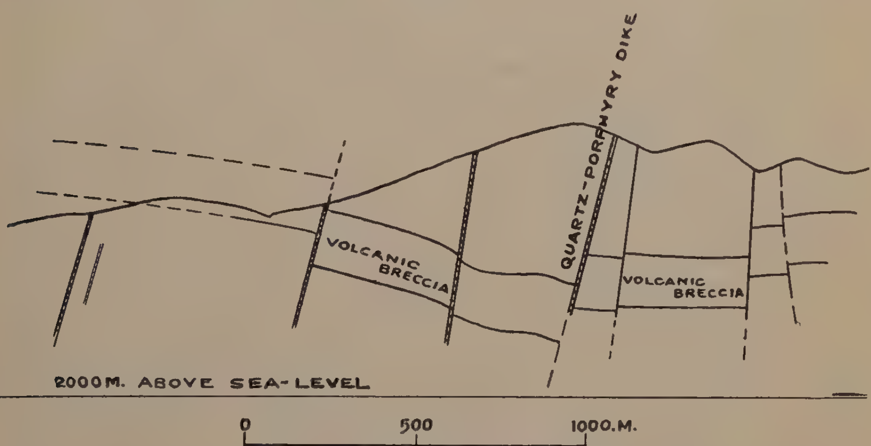


FIG. 4.—VERTICAL SECTION N.14°E., LOOKING WEST.
Approximately at right angles to strike of old northwest fracture system and to strike of folds. Shows: (1) Faults dip complementary to dip of extrusive formations; (2) synthetic faulting, implies vertical force.

Interpretation of this fracture system, using the Cloos conceptions, is to some extent invalidated by the fact that we are not looking at an intrusive massif; we see merely a set of fractures in the cover overlying the assumed buried massif. These fractures may not be upward prolongations of fractures set up in the supposed buried intrusive; but the best guess is that they are. The intrusive is probably large, judging by the extensive area in which shallow intrusives are concentrated. The known large intrusives at Zimapan and Jacala are not deep plutonic masses; they range from monzonite porphyry to quite shallow types; none solidified under very thick cover. The thickness of the cover at Pachuca likewise was probably small compared with the area of the intrusive; many of the major fractures set up in the latter probably worked their way clear to the surface.

If the fracture system typified by the fractures of Fig. 4 is the upward extension of a system set up in the supposed buried intrusive, Nos. 2

and 3 of the Cloos fracture classes (flat-lying normal faults and marginal fissures) may be rejected because they do not fit. Fractures of both these types are relatively flat-dipping, and are characteristically normal faults. The fractures of the N.75°W. system in the Pachuca district dip steeply and some are reverse faults. They suggest class 1, the cross joints. An intrusive massif being stretched in a direction roughly north-south, possibly because the region was subjected during the emplacement of the magma to a nearly east-west compressive stress, would set up cross joints perpendicular to the stretching and roughly parallel to the compressive stress. Such joints would be oriented about as are the fractures of the N.75°W. system. The vertical forces necessary to produce the synthetic displacements shown point strongly to magmatic stresses below; they suggest an ability to push upwards persistently inherent in the molten portion of the magma lying beneath the solidified upper portion in which the supposed cross joints were formed.

If the gentle folding of the andesites was caused by vertical forces exerted by the magma, that folding may well reflect the shape of the upper surface of the buried massif. In that case the dip of the fractures, complementary to that of the extrusives, would be easily explained. (For comparison, see Balk[4].)

These ideas gain added plausibility from the manifest permeability of the fractures of this system toward dikes, a permeability readily explainable had the fractures lain parallel to a persistent east-west compressive force. Dikes oriented at right angles to this supposed force are nearly nonexistent (Fig. 2). The fact that the fracture system kept developing during the epoch of intrusion of the various dike types accords with the notion of a magma emplacing itself, and differentiating, under a regional stress that kept the cross joints squeezed open and continued to form new joints.

The roughly east-west compression called on above accords in age with Thayer's Miocene diastrophism. Study of the sequence of geologic events shows that the time interval between the extrusion of the (Miocene) rhyolite flows and tuffs and the intrusion of the quartz-porphyry dikes (supposedly intruded while the compression still kept the cross joints squeezed open) was relatively small, so that the dikes, too, are probably of Miocene age. It is true that the compression that might have formed the cross joints at Pachuca must have differed somewhat in direction from the northeast-southwest compression that appears to have formed the sharp folds northwest of Pachuca, since the compression at Pachuca must have acted roughly east-west; but it will be shown that certain rhyolite necks in Real del Monte solidified under a compressive stress that acted in a direction lying between northeast-southwest and east-west; i.e., in a direction intermediate between that of the compression northwest of Pachuca and that supposed to have formed the

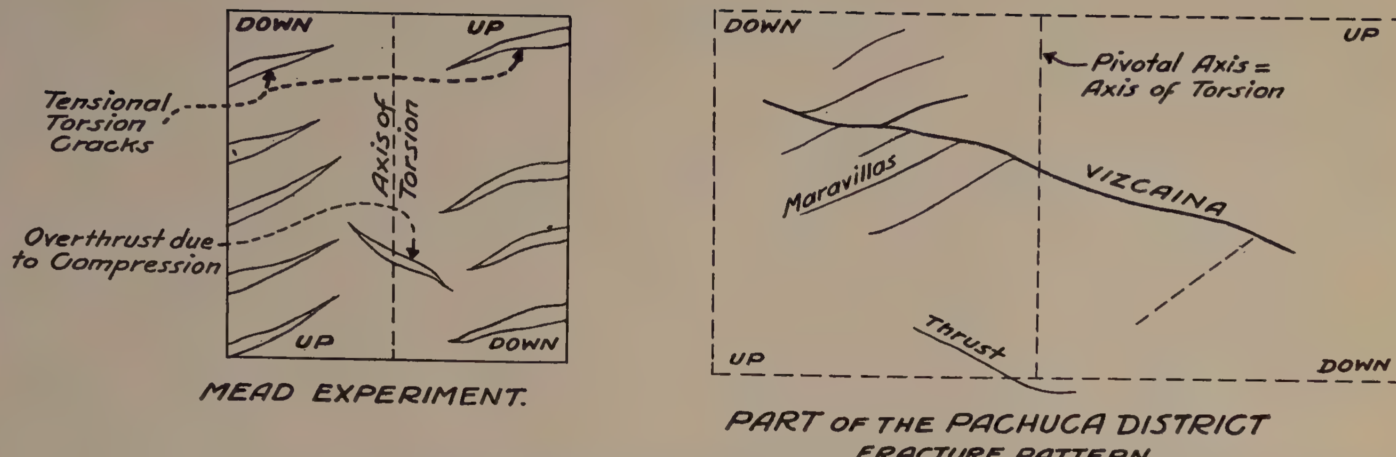


FIG. 5.—COMPARISON OF PORTION OF PACHUCA DISTRICT FRACTURE PATTERN WITH MEAD EXPERIMENT, USING PARAFFIN-COATED RUBBER SHEET AND COMBINING TORSIONAL WARPING WITH BENDING.

original cross joints. It is therefore quite probable that the regional compressive stress, acting at first nearly east-west, swung around to a direction approaching northeast-southwest toward the close of the tectonic activity.

The two principal faults of the N.75°W. system, the Vizcaina (Fig. 2) and the Arévalo, 8 km. north of the Vizcaina, are pivotal faults. Both dip south predominantly, although the Vizcaina locally turns over and dips steeply north. The pivotal movement is the same on both: for the Vizcaina, the axis of rotation lies between Pachuca and Real del Monte; for the Arévalo, the axis is nearly due north of this point. The north block of each fault went relatively down west of the axis of rotation, and relatively up east of that axis. Such movements suggest torsional warping, probably ascribable to forces connected with the emplacement of the buried intrusive. Certain northeast and southwest branches of the Vizcaina (Figs. 2 and 5) accord with the torsional tension fractures obtained by Mead[6] in one of his experiments on torsion. Even the equivalent of the central fault obtained by Mead, and described by him as a thrust fault, is present in the south part of the Pachuca district. This pivotal faulting occurred at an early stage, as a supposed torsional branch of the Vizcaina carries a quartz-rich dacite dike.

Summary.—To summarize, the earlier history of the Sierra de Pachuca is one of extrusion, warping, gentle folding and intrusion, and includes the formation of a parallel fracture system, the fractures of which are the loci of nearly all the dikes of the district. Many of these fractures are faults whose primary movements, as determined by both field evidence and theory, were largely up or down their dips, with little lateral movement parallel to the fault strikes.

Field evidence shows that this fracture system, including the torsional fractures and the thrust mentioned above, is older than any other system that will be described. Examples of such evidence are: (1) the fact that this system, alone, carries the two older dike types (quartz-rich dacite and normal dacite) and 90 per cent of the quartz-porphyry dikes; (2) fractures of other systems invariably fault the older dikes, and usually fault the quartz-porphyry dikes; and (3) the displacements, up or down the dip, on the N.75°W. fracture system may be shown to have occurred, in all probability, prior to the formation of fractures of other systems. (See below.)

After this older fracture pattern had been formed and the dip-faulting connected with it had been completed, and after intrusion had nearly ceased, a different type of fracturing and faulting took place in the Real del Monte area. This later fracturing seems closely linked in time to the mineralization. It forms the main subject of this paper; but since an attempt will be made to show that the later fracturing was a consequence

of the events already described, a somewhat detailed account of these events has been unavoidable.

Mineralization.—A number of the older fractures, as well as many of the younger ones to be described, were mineralized at about the end of the second period of rhyolitic intrusion. Silver-bearing veins were formed. Their chief gangue is quartz. Sphalerite and galena are fairly abundant in places. Vein pyrite is plentiful in the north part of the Real del Monte area only. Primary silver occurs almost entirely as argentite. The evidence points to but a single period of mineralization but a somewhat protracted one, made up of distinct stages. Mineralization in the Real del Monte area and its relation to structure is discussed in detail a little later in the paper.

The Real del Monte Area

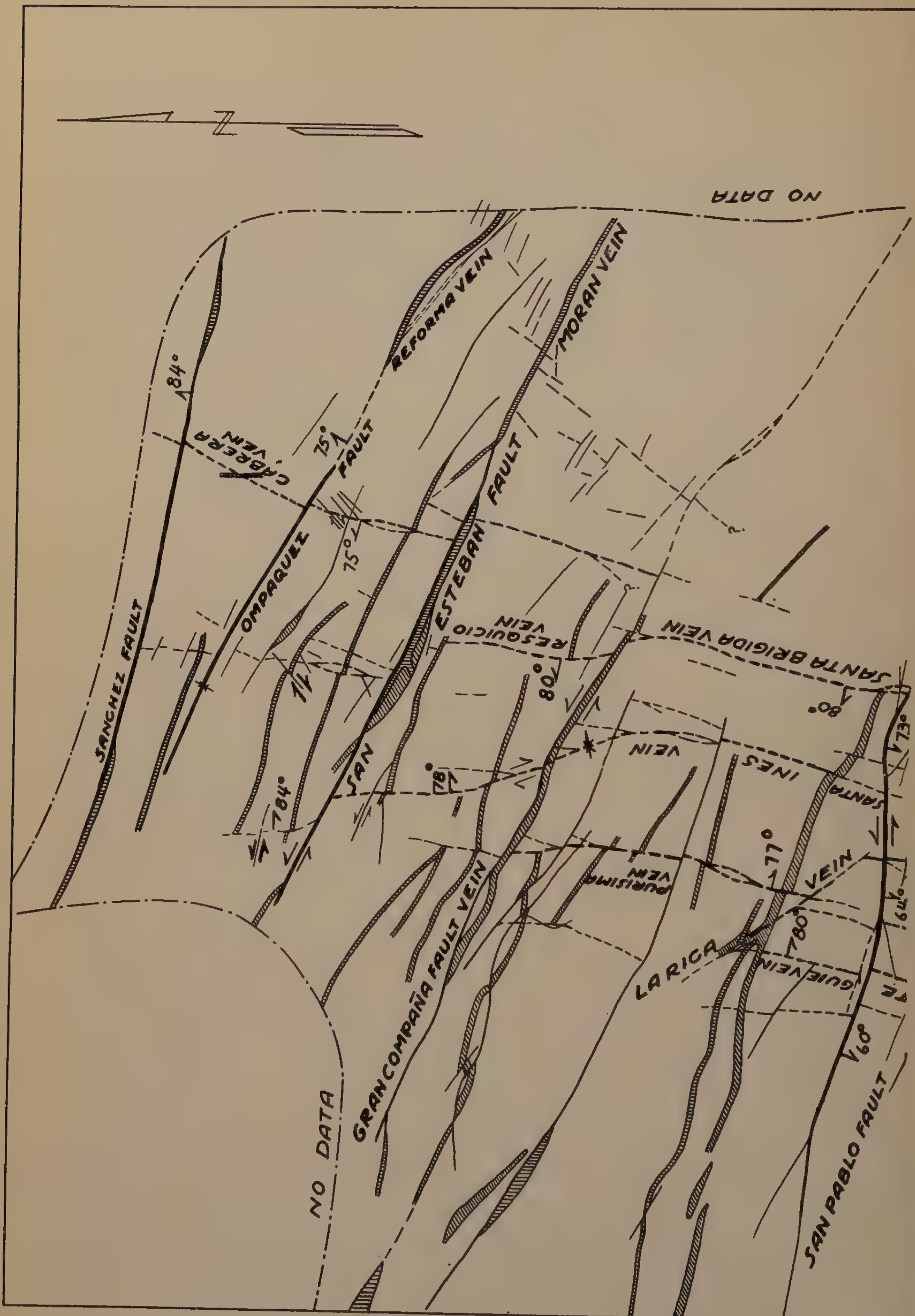
Fig. 6 is a horizontal projection of the main fractures and dikes of the area. The datum plane for the projection lies about 400 m. below the surface. The country rock at this horizon consists of the older (augite andesite) extrusive series, mainly massive flows. The N.75°W. fracture system described above is strongly represented; it is followed by nearly all the dikes. The picture is complicated, but the facts given below, visible on the plan or derived from field and office study, seem significant.

1. The north-south fractures form a well defined belt and show a fairly even spacing. They are nearly all faults or fault zones of small displacement, but are called fractures to differentiate them from the more continuous and usually stronger faults of the other systems; these "fractures" are important mostly because they became veins. The north-south fractures dip steeply. They are zones of brecciation and parallel sheeting. In some places these were merely zones of potential fracturing and are locally discontinuous.

2. Within the belt of north-south fractures, the Gran Compaña and Vizcaina fault-veins show marked irregularities in strike and dip; they are much more regular east and west of the belt. Further, the Gran Compaña and Vizcaina have in general flatter dips than the other faults of their system.

3. Where faults of the N.75°W. system cross the belt of north-south fractures, they commonly show strong gouge, more than they show east of the belt. To the east, they carry in places quartz and ore; within the belt and for some distance west of it, they are mainly barren faults rich in gouge. Of this system, only the irregular Gran Compaña and Vizcaina faults carry quartz and ore within the north-south fracture belt.

4. The main faults of the N.75°W. system, and many of the lesser faults, commonly, but not always, fault the north-south fractures.


NO DATA
REFORMA VEIN
MORAN VEIN
84°
CABRERA VEIN
75°
FAULT
75°
OMPAQUEZ
SAN ESTEBAN FAULT
RESQUICIO VEIN
SANTA BRIGIDA VEIN
SANCHEZ FAULT
80°
80°
78°
84°
SANTA INES VEIN
73°
SAN
77°
PURISIMA VEIN
VEIN
78°
64°
LA RICA
GUIE VEIN
GRANCOMPAÑA FAULT VEIN
60°
NO DATA
SAN PABLO FAULT

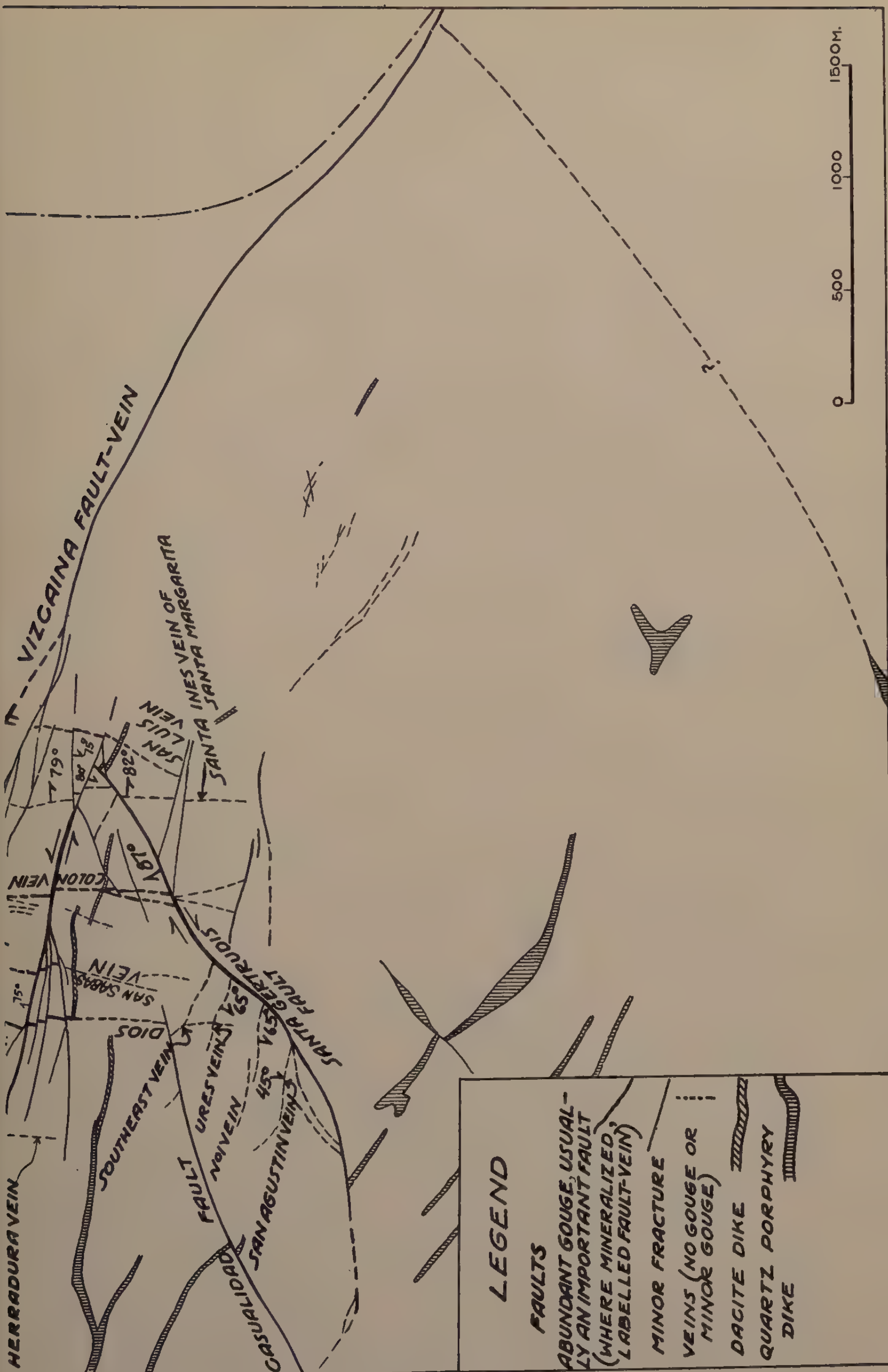


Fig. 6.—Horizontal projection of veins, faults and dikes of Real del Monte area.

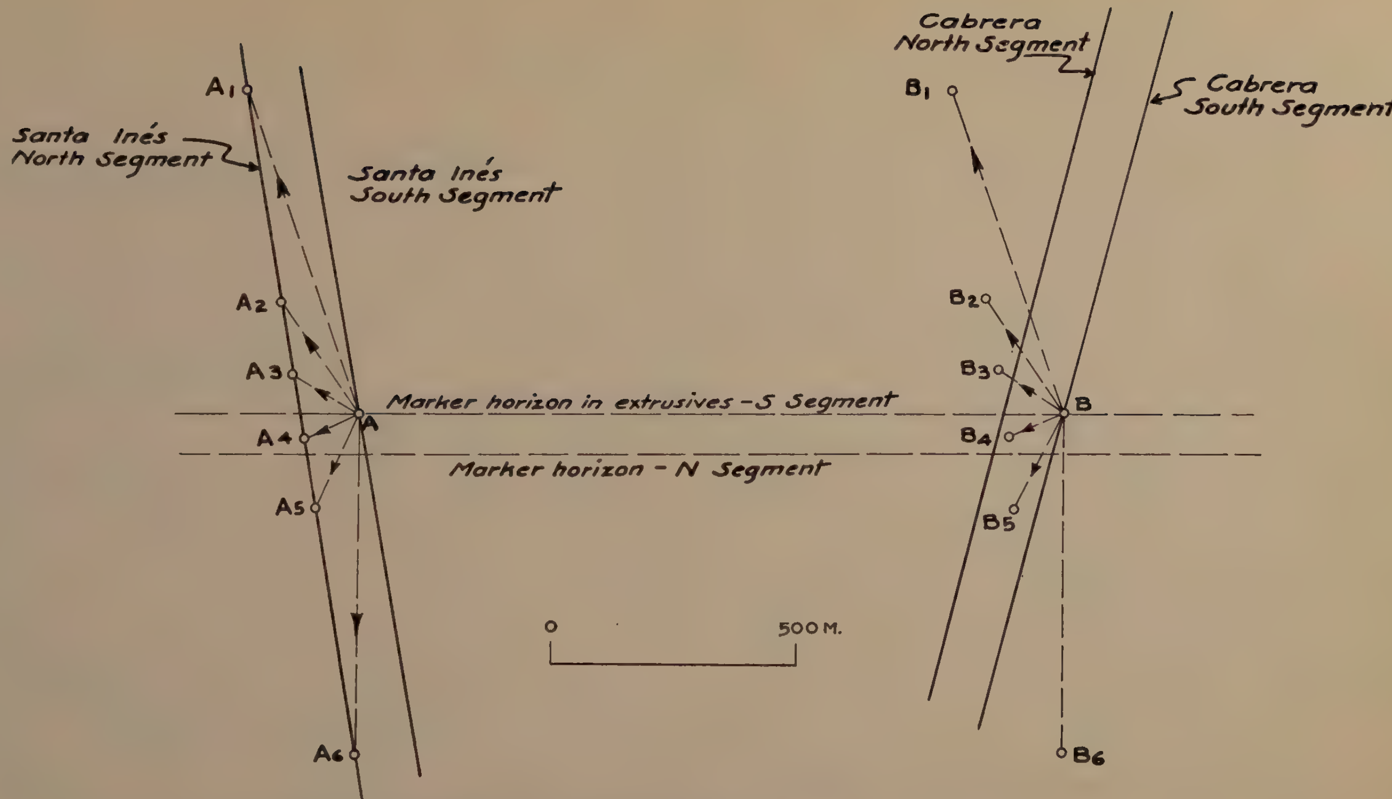


FIG. 7.—PROJECTION NORMAL TO SAN ESTEBAN FAULT PLANE, LOOKING NORTH.

A, a point on trace of south segment, Santa Inés north-south fracture.

A_1, A_2, etc., possible positions a point touching *A* before displacement might have been moved by displacement. Lines A-A_1, etc., represent, in principle, all possible directions of net slip except horizontal. *B* is point on trace of south segment of Cabrera north-south fracture. Same net slips drawn for *A* have been drawn for *B*. No direction of net slip accounts for observed displacements of Santa Inés and Cabrera fractures except horizontal. But horizontal movement indicated, 120 m., could not have produced observed displacement of extrusive formations.

Where this occurs, the horizontal displacement is always with the north side moving west. On the other hand, north-south fractures frequently fault fractures of the N.75°W. system; and when this occurs the displacement is always with the west side moving north. Here are two mutually displacing sets of faults.

5. South of the Vizcaina fault-vein the N.75°W. system is much less strongly developed than north of it. On the south the fractures tend to be irregular and discontinuous. A strong northeast fault, the Santa Gertrudis, appears in this area, accompanied by one or two others similarly oriented. These faults invariably displace the north-south fractures they intersect, and the direction of horizontal displacement is with the south side moving west.

6. The types of displacement just described (north side west on the N.75°W. faults, west side north on the north-south fractures, south side west on the northeast faults) all took place very nearly in a horizontal plane. The overwhelming majority of the striae on the faults of all three systems pitch within 20° of the horizontal. Analysis of a number of the fault movements confirms the evidence of the striae. Thus Fig. 7 is a projection on the San Esteban fault surface: the plane of the paper represents the fault surface. The traces on that surface of the faulted segments of the Santa Inés and Cabrera north-south fractures are shown, as well as those of a marker in the extrusive formations, which here lie practically flat; that is, complementary to the dip of the vertical San Esteban, a member of the N.75°W. synthetic fault system. Plainly, the only single-stage movement that explains the observed displacements of the Santa Inés and Cabrera fractures is horizontal movement. But such movement could not have caused the displacement of the extrusive formations shown in the figure. Two periods of faulting explain the facts most simply. It has already been shown that the synthetic nature of the movements on the faults of the N.75°W. system implies vertical forces and consequently suggests directions of movement largely up or down the dips. Such movement would readily account for the displacement of the rock formations shown. If the north-south fractures had been present during this faulting, they would have been displaced by it in such a way that later horizontal faulting (shown to have occurred by the abundant late horizontal striae) would have displaced the Santa Inés 140 m. the north side moving west, and the Cabrera 90 m., instead of the actual displacement of both, 120 m. It is reasonable to suppose that the north-south fractures were not in existence during the supposed dip-faulting. Field study lends weight to this reasoning. A quartz-porphyry dike was intruded along the San Esteban fault. It is everywhere exactly along the south side of the fault surface. East of Real del Monte, where horizontal movement has been small but where the vertical displacement of the extrusives is still evident, the dike is practically

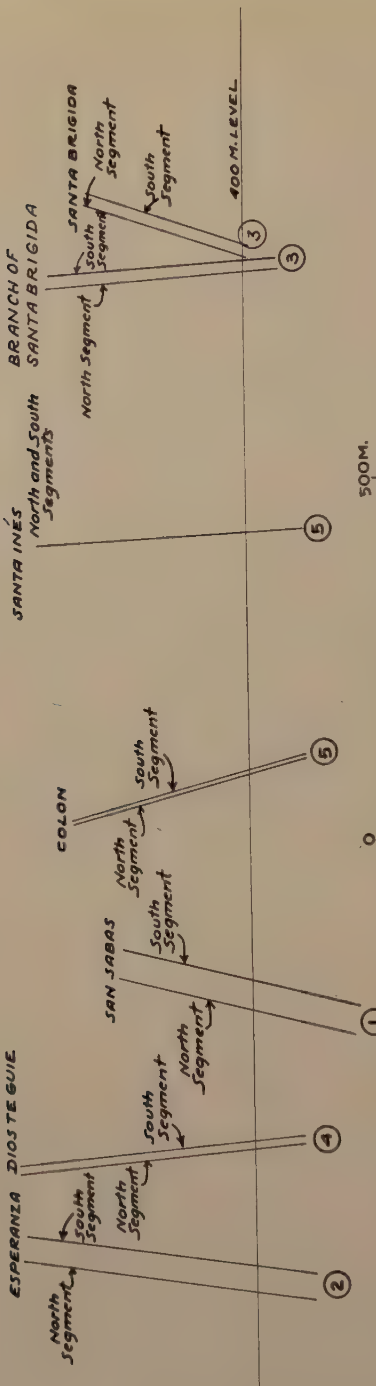


Fig. 8.—Projection normal to Vizcaina vein, looking north. Showing offsets of north-south veins. Numbers in circles indicate relative age of vein fractures.

unbroken; within the belt of north-south fractures it is shattered and in places shows horizontal striae. The dike has been subjected to horizontal movement only; obviously it must have come in after movement of any other type, which means after any movement that could have produced the displacement of the extrusives shown in the figure. Since the north-south fractures are demonstrably later than the quartz-porphyry dike, it follows that they, too, were not in existence during the earlier faulting.

This analysis has been given in detail because it covers one of the clearest cases of movements, and because nearly similar reasoning may be applied to a number of other N.75°W. faults, to the Santa Gertrudis fault, and to movements on the north-south fractures.

7. Confirmation of the conclusion indicated by the discussion above (namely, that the north-south fractures are later than most of the N.75°W. fractures) is supplied by the fact that the north-south fractures carry, if any, the latest type of dike (quartz-porphyry) and that in only one part of the area; and even there they cut across and often fault all older types of dike (quartz-rich and normal dacite) and nearly all transverse quartz-porphyry dikes. This suggests that the quartz-porphyry intrusion was almost at an end at the start of formation of the north-south fractures.

Thus, although the N.75°W. fractures frequently fault the north-south fractures, the former are, for the most part, the older. Their faulting of the north-south fractures was by late horizontal movement on faults whose earlier movements were largely up or down their dips.

8. The north-south fractures were forming at the time of the horizontal movements on the N.75°W. faults. Fig. 8 is a projection on the Vizcaina fault surface. As with the San Esteban, the throw of the rock formations by the Vizcaina (300 m.) occurred prior to any displacements of north-south fractures. But in the Vizcaina the amounts by which these fractures are displaced vary from 0 to 40 m. On the east, the Santa Brigida fracture, and its branch (thought to be of contemporaneous origin), dipping toward each other at different angles, are both displaced 13 m., the north side moving west, by the Vizcaina. Striae on that fault confirm the implication of horizontal movement derived from the geometry. Near the center of the figure the Santa Inés fracture is not displaced at all, the Colón only slightly. On the west, the San Sabas and Esperanza fractures are displaced about 40 m., all displacements being with the north side moving west. The idea that all the north-south fractures were there before horizontal movement on the Vizcaina started, and that the differences in horizontal displacement were caused by variations in the intensity of the movement on the Vizcaina, is untenable or at best improbable. It seems logical to think that the north-south fractures were forming while the horizontal movement on the Vizcaina was going on. The north-south fractures displaced most were there earliest;

those undisplaced were formed, near the Vizcaina at least, only when the horizontal movement on that fault had practically ceased. Evidence contributing to belief in this idea is given below.

9. Both the San Pablo and the Santa Gertrudis faults displace north-south fractures, such as the Santa Inés vein of Santa Margarita and the Colón vein, that cross, but are not displaced by, the Vizcaina, or are only very slightly displaced. The San Pablo fault displaces these two veins the same distance, 40 m., the north side moving west; the Santa Gertrudis displaces each of these veins the same distance, the south side moving west also, as it happens, about 40 m. The simplest explanation is that these movements on the San Pablo and Santa Gertrudis faults occurred later than the horizontal movement on the Vizcaina; the north-south fractures just mentioned, which formed only when movement on the Vizcaina had practically ceased, were in existence at the start of the movements on the San Pablo and Santa Gertrudis faults. Neither the San Pablo nor the Santa Gertrudis fault carries a dike; the latter displaces a quartz-porphyry dike, the youngest type. There is, therefore, evidence contributing to the idea of late age for these faults.

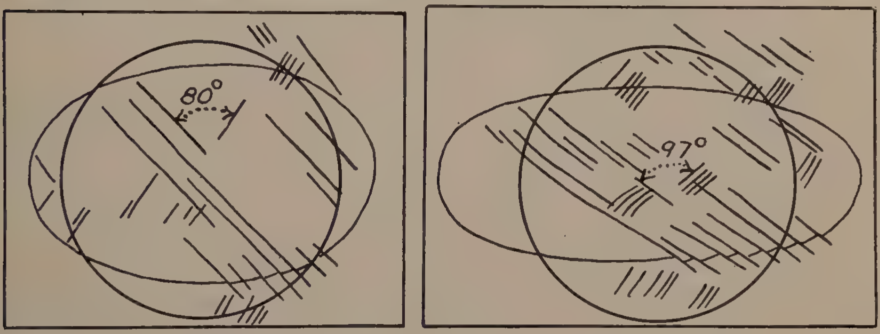
10. Not only do the north-south fractures in the north end of the district carry mostly the early vein minerals, but, as noted below, it is there only that they carry, for short distances, quartz-porphyry dikes. It has been stated that this intrusion immediately preceded the mineralization, overlapping it slightly. The suggestion is strong that some of the north-south fractures started forming in the north part of the area, either at the beginning of the mineralization or just prior to it, and worked their way, in general, southwards.

As related to the fracture pattern, a word may be said here regarding mineralization. Massive pyrite is the earliest sulfide throughout the area; it is often contemporaneous with an early type of quartz, and is followed by coarse sphalerite, coarser grained than that associated with silver. These early minerals are usually nearly barren of silver. In the north part of the Real del Monte area the barren iron-zinc type of vein filling, and the silver-bearing, lead-zinc type, are found together, and the latter type is clearly later than the former. The veins of northern Real del Monte, both the north-south fractures and the sparsely mineralized N.75°W. faults, show this early iron-zinc surge predominating strongly over later type of vein filling. The early vein minerals decrease in amount progressively from north to south, with local exceptions. The relative amount of lead increases and silver orebodies become more plentiful. Polished-surface study shows that much zinc was deposited before any silver, and that after deposition of the silver began the relative amount of lead (compared to zinc) kept increasing to the end of the main period of deposition of base sulfides. Silver kept on depositing for some time after the bulk of the base sulfide deposition had ceased.

Mechanics Suggested to Explain the Late Fracture Pattern of the Real del Monte Area

So far, in the description of the Real del Monte area, the assertions made, and conclusions drawn, are believed to be founded on fact. They contain no more of speculation than is usually found in attempts to decipher the geologic record. Most geologic "proofs" are really no more than well founded opinions. These are no exceptions to the rule.

But in attempting large-scale exploration, beyond the more or less explored portions of a district, a broader comprehension is necessary than that contained in the facts given above. It is extremely desirable to know, before directing work into a totally unexplored area, something about how the fractures may lie in that area and along what set of fractures ore may most reasonably be hoped for. Simple extrapolation from



a *b*

Fig. 9.—Deformation of clay cake, looking down on cake.
a. Stage at which shear planes first appear.
b. Later stage.

known fracture patterns seldom works except for short distances, sometimes disconcertingly short. If the mechanics of the fracturing and the vein formation can be worked out for the district as a whole, it is obvious that a start may often be made in understanding an unexplored area before any actual mine exploration work is done. Only where such an understanding exists can exploration be intelligently started.

It was with such ideas in mind that I attempted to work out a comprehensive theory to explain the fracturing and vein formation of the Real del Monte area. After a considerable study of the experiments described by others, and comparison of these results with field observations, I decided that the experiments on faulting and fissuring conducted by Hans and Ernst Cloos, using wet clay as a medium, were most in accordance with field experience, therefore probably the most reliable. The experiments of Hans Cloos that seemed most applicable to the problem at hand are summarized in Figs. 9, 10 and 11. The following descriptions are summarized from his works.

Experiment 1.—In the experiment shown in Fig. 9[7,8] a cake of fine clay about 10 cm. thick, kneaded in water, was supported on a rectangular wire netting. The netting was changed in shape to a rhomb, and the clay cake participated in the deformation. It did not matter *how* the netting was deformed, whether by pushing or pulling (compression or tension), or by holding one edge of the rectangle firm and displacing the opposite edge in a direction parallel to itself. No matter what kind of force was used, the same fractures resulted for a given deformation. The clay surface was smoothed before the experiment, and a circle was drawn upon it. Then the support was lengthened east-west, correspondingly shortened north-south, by any of the methods mentioned. At first the clay behaved plastically, slightly deforming the circle into an ellipse, without fracture. Soon, however, two sets of shear fractures appeared, making an initial angle with each other of from 75° to 80° (in later, similar experiments as small as 63°). The smallest axis of the ellipse bisected the acute angle between the shear sets. The shear planes stood perpendicular to the surface of the picture; i.e., vertically.

The experiment was so conducted that there was no deformation of the clay cake in a direction normal to its horizontal surfaces; that is, there was no thickening or thinning of the cake. All deformation was horizontal. The circle, deformed into an ellipse, represents the section of the ellipsoid of deformation that contains the major and minor axes of strain. The intermediate axis stands vertically and is identical with the line of intersection of the two sets of shear planes. If the deformation is continued, the acute angle between the shear planes widens, becoming obtuse; displacements occur along the planes in the general direction of the major movement.

Experiment 2.—To illustrate antithetic and synthetic faults, H. Cloos performed the experiments[7] illustrated in Fig. 10. A clay cake similar to the one described rested on a divided foundation, the two blocks of which were pulled away from each other, permitting the cake to sag between the retreating blocks. First, the cake bends and the unsupported portion sinks; then two fault zones form, bounding a wedge-shaped block of the cake, and permit the unsupported portion to sink further as a graben. But the movement is unsymmetrical. The right side is a hinge zone, around which the graben block has rotated in a counterclockwise direction. The movement produces, therefore, on the left, normal, synthetic faults dipping toward the sunken block; on the right side the faults dip away from the sunken block, but the fault surfaces are bent to the left. These are antithetic faults. They are also shear planes, as shown in the figure, where the ellipses are deformed from circles drawn on the surface of the clay before the start of the experiment. The other shear set on this right-hand side is represented by the strong left-dipping fault touching the edge of the right-hand base block and failing

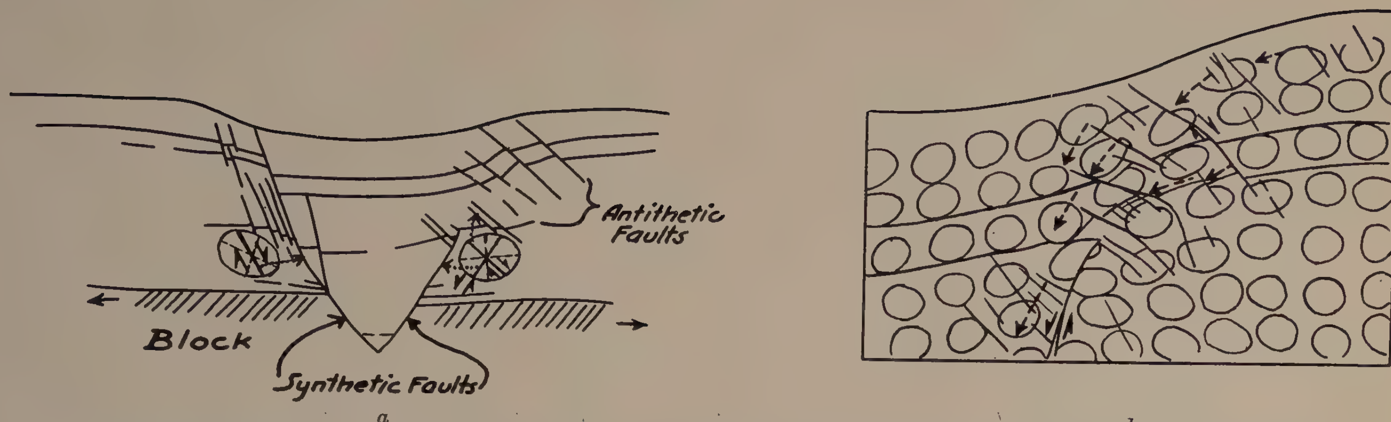


FIG. 10.—EXPERIMENT WITH CLAY CAKE, ILLUSTRATING ANTITHETIC FAULTS ACCOMPANYING UNSYMMETRICAL GRABEN FAULTING.
a. Side view of graben.
b. Enlargement of portion of right-hand side of *a*. Arrows show actual movement of clay.

to break through to the surface. Study of Fig. 3 will show why this fault is synthetic, the faults of the other set antithetic. The acute angle between the two sets of shear fractures (initial angle) is bisected by the direction of shortening; but rotation changes it to an obtuse angle. The same thing occurred in the experiment of Fig. 9, through continued deformation.

On the left-hand side of the graben, the major right-dipping fault is a shear plane, and the downward movement of its hanging wall, together with the resistance of its footwall, would have formed a clockwise shearing couple, had there been any real resistance of the footwall. The movement on the major fault is synthetic; but the antithetic set of shear planes was not formed, evidently because the footwall offered practically no resistance to the movement of the hanging wall; shearing *stresses* in the clay were small or nonexistent.

Fig. 10*b* is an enlargement of the area of antithetic faults of the right-hand side of Fig. 10*a*. The true nature of the deformation is shown by the ellipsoids, deformed from circles drawn before the start of the experiment. The dashed lines with arrows show the true paths of particles in the clay, and the lengths of the lines indicate the distances traveled in a given time. The true movement is shown to be nearly at right angles to the antithetic fault planes, just as in Fig. 3*a*; also, the true movement is not parallel to the major axes of the ellipsoids, although it usually shows a tendency toward parallelism.

Experiment 3.—Fig. 11 shows two successive stages in a modification of the experiment just described; it was performed to imitate a set of antithetic faults in Alsace[9]. The pictures are reversed from the way they were published to facilitate comparison with Figs. 10 and 12. The side of the subsided block is shown corresponding to the right-hand side of Fig. 10*a* and showing rotation and antithetic faults. But in this experiment the left-dipping major shear plane (representing the synthetic set) is more continuously developed than in experiment 2; the antithetic shear set forms as a group of fractures subsidiary to the major fault; they are complementary or secondary shear planes.

In experiment 2 (Fig. 10) the rotation that formed the antithetic displacements was due to the uneven settling of the graben; in experiment 3, the rotation was caused by one or more irregularities in the major fault that impeded free slipping of the graben down that fault. A sharp salient into the hanging wall appears about halfway down the dip of the major fault in Fig. 11*a*; while an antithetic shear plane forms its "northeast" border, it looks from the original photograph as though the antithetic plane had merely sheared off an edge of a pre-existing, irregular salient. Fig. 11*b* shows that continued sinking of the graben had planed off this salient, and that, once the salient had gone, the antithetic faults ceased forming, although synthetic faults (parallel to the major fault) continued to develop.

In this experiment, as in that of Fig. 10*a*, a graben block was caused to sink into the space left between two foundation blocks that were pulled apart. Fig. 11*a* is a vertical section; but if we think of it as a plan, the same results would have been obtained had the two foundation

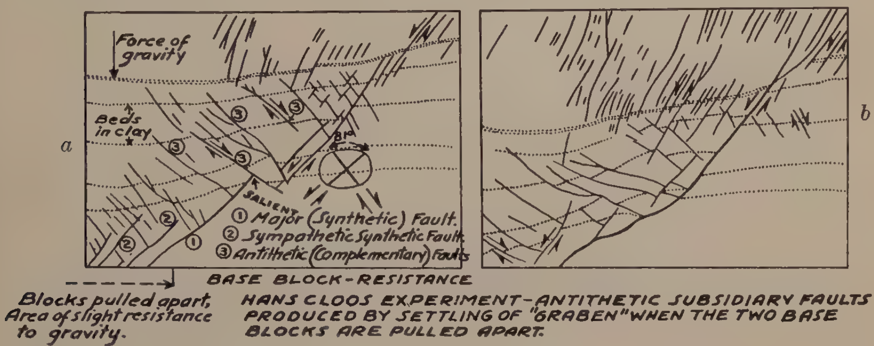


Fig. 11.

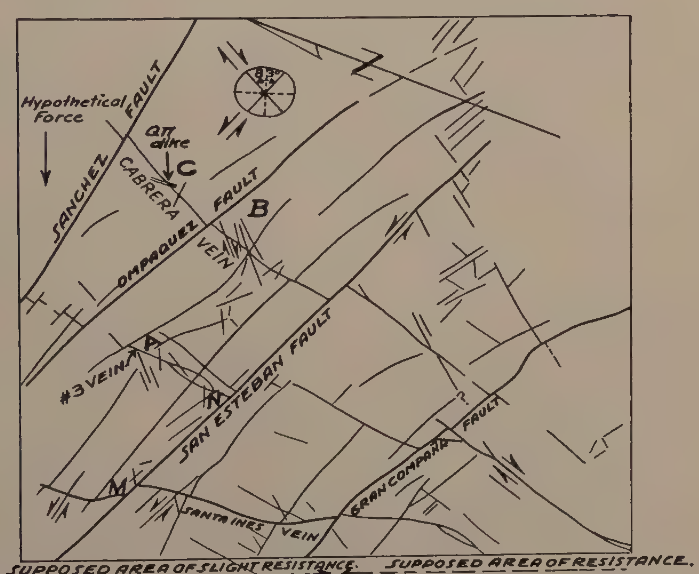


Fig. 12.

Figs. 11 and 12.—Clay experiment illustrating synthetic and antithetic faults, compared with northern Real del Monte area.

Fig. 11*a*. Stage in clay experiment showing moderate rotation of antithetic faults.
Fig. 11*b*. Later stage, with strong rotation of antithetic faults.
Fig. 12. Portion of northern Real del Monte area oriented for comparison.

blocks been replaced by two strong but movable obstructions placed next the clay mass; if these obstructions had been pulled apart, and the clay mass had been shoved horizontally into the growing free area between the obstructions, the results of Cloos' experiment could, with a little

manipulation, be duplicated. The only difference would be that in the actual experiment the sole active force was gravity; in the hypothetical experiment, the force is horizontal and the action of gravity must be prevented.

Stages Deduced for Real del Monte.—Fig. 12 is a plan of the northern Real del Monte area, oriented for comparison with the experiment of Fig. 11.

The resemblance must be at once apparent. The successive stages, derived from the two stages of the experiment, and deduced by analogy for Real del Monte, are as follows:

1. For the experiment, formation of the major fault (the major synthetic shear plane), and probably at very nearly the same time, formation of a zone of sympathetic faults of the same system, in areas where slipping along the major fault was unhindered. These are analogous to the sympathetic faults of the synthetic system developed on the left-hand side of Fig. 10*a*. Here, too, slipping was free on the major fault, and only one set of shear planes was developed.

In Real del Monte, a number of planes seem to correspond to the synthetic system of the experiment; e.g., the Sanchez, Ompaquez, San Esteban and Gran Compaña faults. Most or all of these are old faults belonging to the preëxisting N.75°W. fracture system; they would appear to have served as major shear planes because they were properly oriented in relation to the stress that caused the shearing.

No sooner had slipping started on the major fault of the experiment than it encountered resistance at the salient. This set up rotational stress and shear planes belonging to the set complementary to that represented by the major shear plane. That this complementary set (the right-dipping faults in Figs. 11*a* and 11*b*) formed early is shown by the fact that while many of its faults displace faults of the synthetic set, they are as often displaced by faults of the latter set.

In the experiment, the initial acute angle (81°), between the right-dipping or secondary shear planes and the major fault was bisected by the direction of shortening (shown by the ellipsoids in Fig. 11*a*). In Real del Monte, the north-south fractures, such as the Cabrera, Santa Inés, etc., appear to correspond to the right-dipping planes of the experiment, and their corresponding initial angle with the major planes was about 83°, judging by north-south fractures that appear to have suffered the least amount of the rotation described below. Note that in Real del Monte the N.75°W. faults, or major shear set, and the north-south fractures are two sets of mutually displacing faults, and that the displacements are in the same directions as in the experiment.

In the experiment, the stronger right-dipping shear planes appear to have been localized by the salient. In northern Real del Monte the counterpart of the salient is probably furnished by the sharply irregular

stretch on the Gran Compaña fault, already described as lying within the belt of north-south fractures.

The right-dipping shear planes in the experiment are more strongly developed in the active than in the passive block. In northern Real del Monte, north-south fractures formed extensively north of the Gran Compaña fault; i.e., continuing the analogy with the experiment, in the active block the ground moved from east to west, not vice versa. But in Real del Monte all the blocks between the N.75°W. faults seem to have moved from east to west, although through distances decreasing in amount as we pass from the northernmost block toward the southernmost; all the blocks were more or less active, and many of the north-south fractures broke clear across them, sometimes hooking up with planes set up in adjacent blocks, and sometimes not doing so.

In the experiment, the planes sympathetic with the major, left-dipping fault kept forming while the graben sank, in the areas removed from the influence of the salient. In northern Real del Monte, while a number of the corresponding planes were pre-existing features, being members of the old N.75°W. fracture system, it is likely that a number of new ones, striking also roughly N.75°W., were formed by the horizontal stress. This is hypothesized because many faults of this orientation carry no dikes; further, they fault and are faulted by minor north-south fractures, and probably are of the same age as the latter.

2. With continued development of the left-dipping, synthetic set in the experiment, a development that kept on after that of the antithetic set had ceased, there was a progressive "rotation" or bending of the right-dipping, antithetic set. The axis of rotation, it seems, was the salient. This fanned out the antithetic planes lying near the salient, and this fanning changed the initial angle between the antithetic planes and the major fault to angles of various sizes. Precisely the same thing is shown in northern Real del Monte. In Fig. 10*b*, where the true paths of the clay particles are shown in relation to the antithetic faults, it is obvious that the rotation of these faults was due to a faster movement of the clay at the left-hand ends of the fractures than at the right-hand ends. In the experiment of Fig. 11, the rotation was due not to the uneven settling of the graben, as in Fig. 10*b*; the cause was a local one, i.e., the impeding of the slipping down the dip of the major fault by the salient. Real del Monte shows, apparently, a similar holding back of the movement, at the irregular stretch on the Gran Compaña. Since this stretch is much larger than the salient, a greater proportion of the antithetic fractures are rotated in Real del Monte than in the experiment.

In spite of its straightness, the San Esteban fault appears to have offered some resistance to the supposed northwestward slipping of its north block, for some rotation of the Santa Inés and No. 3 north-south fractures in that block is apparent, as, for example, at *M* and *N* in Fig. 12.

The Gran Compaña and San Esteban faults carry dacite dikes (Fig. 6); there is therefore no doubt concerning their age: they belong to the old N.75°W. fracture system. But the Ompaquez and Sanchez faults carry none but quartz-porphyry dikes. It has been stated that one or two north-south fractures have been intruded by quartz-porphyry dikes, in this north part of the Real del Monte area. If it is true that the north-south fractures were set up by the shearing stresses now being described,

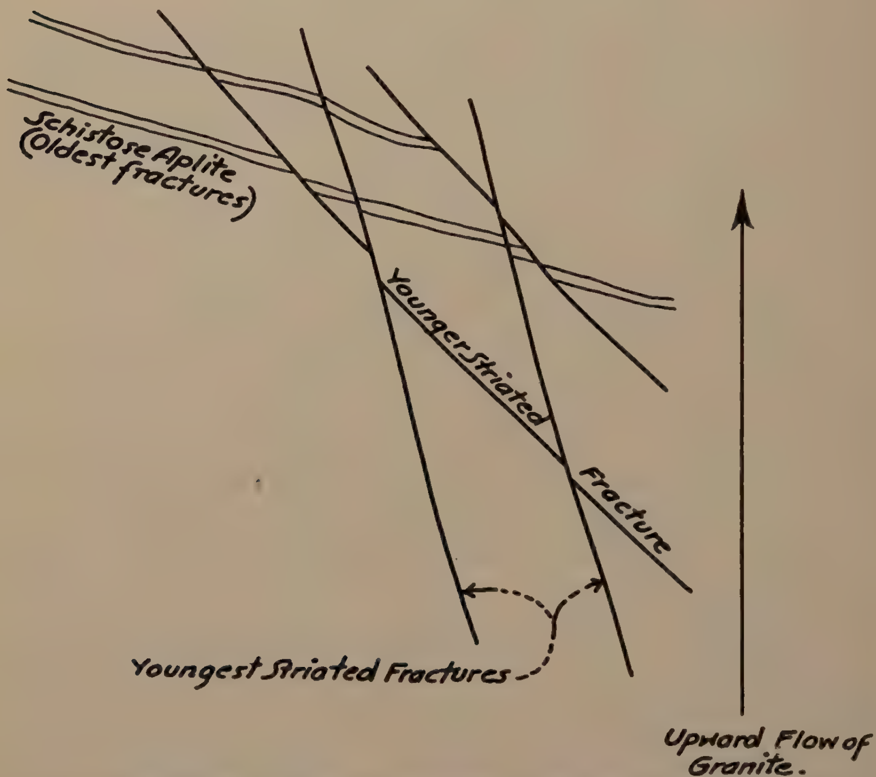


Fig. 13.—Three sets of marginal fissures of granite pluton at Striegau, Silesia, showing progressive rotation.

quartz-porphyry must still have been intruding at the start of these stresses. It would, therefore, not be surprising if a few of the late N.75°W. shear planes (parallel to, but not genetically connected with, the older N.75°W. system) had been locally intruded by quartz-porphyry in favored places, such as where irregularities in strike would tend to cause openings even under strong shearing movement. The Ompaquez and Sanchez faults, show, in truth, only discontinuous stretches of quartz-porphyry along them. The Sanchez fault, especially, looks as though it had been set up by the shearing movements, as seen by comparing its

attitude with that of the late synthetic shear planes near the upper left-hand corner of Fig. 11*b*.

A feature of the Real del Monte area not shown by the experiment is furnished by the northeast minor faults, crossing main north-south fractures obliquely, as at *A* and *B*, Fig. 12, and displacing the north-south fractures in the same direction (the west side moving north) as the north-south fractures themselves habitually displace fractures of the N.75°W. system. These northeast fractures are obviously members of the same complementary shear set as the main north-south fractures, since they make the same initial angle (bisected by the direction of shortening) with the major shear planes as did the main north-south fractures before the latter suffered rotation. The northeast fractures, therefore, are later counterparts of the main north-south fractures. Since they are unrotated and therefore probably late, and are at best only slightly mineralized, they suggest that the shearing stresses continued at least to the end of the period of mineralization in this part of the area.

A nearly perfect analogy to these late antithetic shear planes appears in a paper by H. Cloos[10] describing marginal fissures of glaciers, fault blocks and plutons. Fig. 13, from that paper, shows three different sets of marginal fissures, each set successively more rotated, and the younger set displacing the one next older. The oldest, most rotated set, alone carries aplite dikes. All three of Cloos' sets are present in Fig. 12. At point *C*, a quartz-porphyry dike fills a fracture that is more rotated than the Cabrera vein; the dike represents Cloos' oldest set. His intermediate set is represented by the main north-south fractures, and his youngest set by the northeast minor faults at *A* and *B*.

Study of the vein filling in the part of the area shown in Fig. 12 confirms conclusions drawn from the above study. The vein matter in this part of the area consists largely of minerals and types of minerals that are shown by polished-surface study to have made up the first surge of the mineralization. These consist of a characteristic early-type quartz, coarse zinc blende, coarse galena, and, especially, massive pyrite. The Santa Inés fracture carries these in abundance, but in the northern part of the area referred to here carries only sparse silver orebodies. The Santa Inés, if the comparison with the experiment be a true one, suffered extreme rotation, so that it lay, toward the end of the shearing stresses, practically at right angles to the compressive component of the stress, and parallel with the direction of tension. It is reasonable to suppose that the abundant early minerals, forming very wide bodies even where the Santa Inés is most bent, came in before the strong rotation, when the fracture lay at a much larger angle to the direction of tension, and the tendency for the walls to be pulled apart must have been considerable. On the other hand, the silver solutions, which are known from other evidence to be later in age than the pyrite, coarse blende and minerals

of that group, entered the Santa Inés after most of the rotation, when compression acted strongly against the fracture walls, and silver orebodies could form only in local, exceptionally favorable places.

Confirming this idea, the Cabrera fracture, while its vein filling, too, consists largely of the early minerals, carried a continuous and important silver orebody. Fig. 12 shows that the Cabrera, at the present day, makes an angle of 48° with the direction of tension that prevailed during the shearing stresses. It could never have made less; any rotation would decrease this angle (it decreased it to zero in the case of the Santa Inés), and the Cabrera probably has been rotated slightly. There was, therefore, during the entire part of the period of mineralization during which the Cabrera was in existence, a considerable tendency for the walls of that fracture to be pulled apart. Not only that, but since the Cabrera was oriented correctly after its formation and throughout the remainder of the period of shearing to serve as a complementary shear plane, it was subjected to minor strike faulting admirably adapted to sheet and brecciate its walls. At the time of entry of the silver solutions, the Cabrera was exceedingly permeable. Study of the vein structure shows clearly not only the fact that the walls were being pulled apart during entry of the vein material but also that minor strike faulting was sheeting and brecciating the early quartz and the walls. Indeed, the same applies, to a greater or less degree, to all the north-south veins; that the mineralization came in while the walls were being pulled apart and while minor strike-faulting was going on, is evident after even brief study of the vein structures, and it was the realization of this fact that first directed the writer towards his present conclusions.

The resemblance between the northern Real del Monte area and the experiment of Fig. 11 is so close, even though one is characterized by strike faulting and the other by dip faulting, as to leave little doubt that the late fracturing of northern Real del Monte was brought about as shown. If so, and bringing the analogy with the experiment to its inescapable conclusion, an area of lower resistance than elsewhere, northwest of Real del Monte is required, to correspond to the space between the basement blocks of the experiment; i.e., the space toward which the moving ground was shoved (Figs. 11 and 12). For strict analogy with the experiment, an *increasing* area of low resistance is needed. I believe such an area exists; but since the comparisons with experiments so far made do not cover the whole of the Real del Monte area, the description of the low-resistance area is deferred until the rest of Real del Monte is brought into the picture.

It has been stated that the Vizcaina vein shows a stretch of irregularities in strike and dip like the stretch on the Gran Compaña, and that this irregular stretch lies opposite that on the Gran Compaña (Fig. 6). These irregularities on the Vizcaina probably set up shearing stresses

near that fault, much like those near the Gran Compaña that have been described; such southern shearing stresses aided greatly in the prolongation of the belt of north-south fractures toward the south. The lack of continuity of north-south fractures at or near the Vizcaina is marked, which suggests an area of origin near the Vizcaina for some of the fractures, independent of the area of origin near the Gran Compaña. The Dios te Guie, San Sabás, Colón and Santa Inés of Santa Margarita fractures do not continue with fractures thought to have originated in the north; their angles of intersection with the Vizcaina—i.e., the angle bisected by the direction of shortening (Fig. 12)—are such that, allowing for moderate rotation, they could well be shear planes complementary to the Vizcaina major shear plane, and set up by shearing stresses induced by the impeding of free slipping on the Vizcaina caused by irregularities in the strike and dip of that fault.

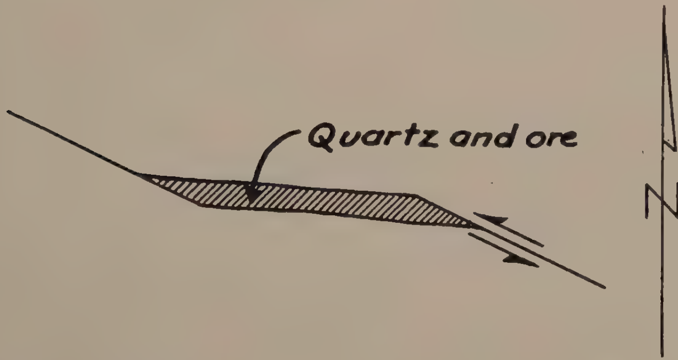


FIG. 14.—LOCALIZATION OF ORE AND QUARTZ ON VIZCAINA FAULT-VEIN.

The fact that rotation of north-south fractures near the Vizcaina was moderate,* and the fact that fractures originating in the north, such as the Santa Inés vein, are not displaced by the Vizcaina, suggest that the Vizcaina horizontal movements, with the attendant formation of some north-south fractures, started later than did the horizontal movements on the north. The vein matter on the Vizcaina within the belt of north-south fractures is shown by field study to be contemporaneous in age with that of the north-south fractures in the Vizcaina area. This vein matter is of younger type than the bulk of the vein filling at the north end of the area. Fig. 14 shows that space was probably made for the Vizcaina vein matter by the horizontal movements along that fault; these movements are thought to have accompanied the formation of a number of north-south fractures that seem to have originated in this

* La Rica vein, shown in Fig. 6, is not the rotated continuation of the Colón. It belongs to the older fracture system. It behaved, however, like an extremely rotated north-south fracture: it carried narrow ore, and it displaces slightly dikes and fractures, the west side moving north.

area, as I have stated. Field evidence indicates that all was one continuous process—horizontal movement on the Vizcaina, formation of these north-south fractures, and the entry of vein matter during these events into the Vizcaina and the north-south fractures.

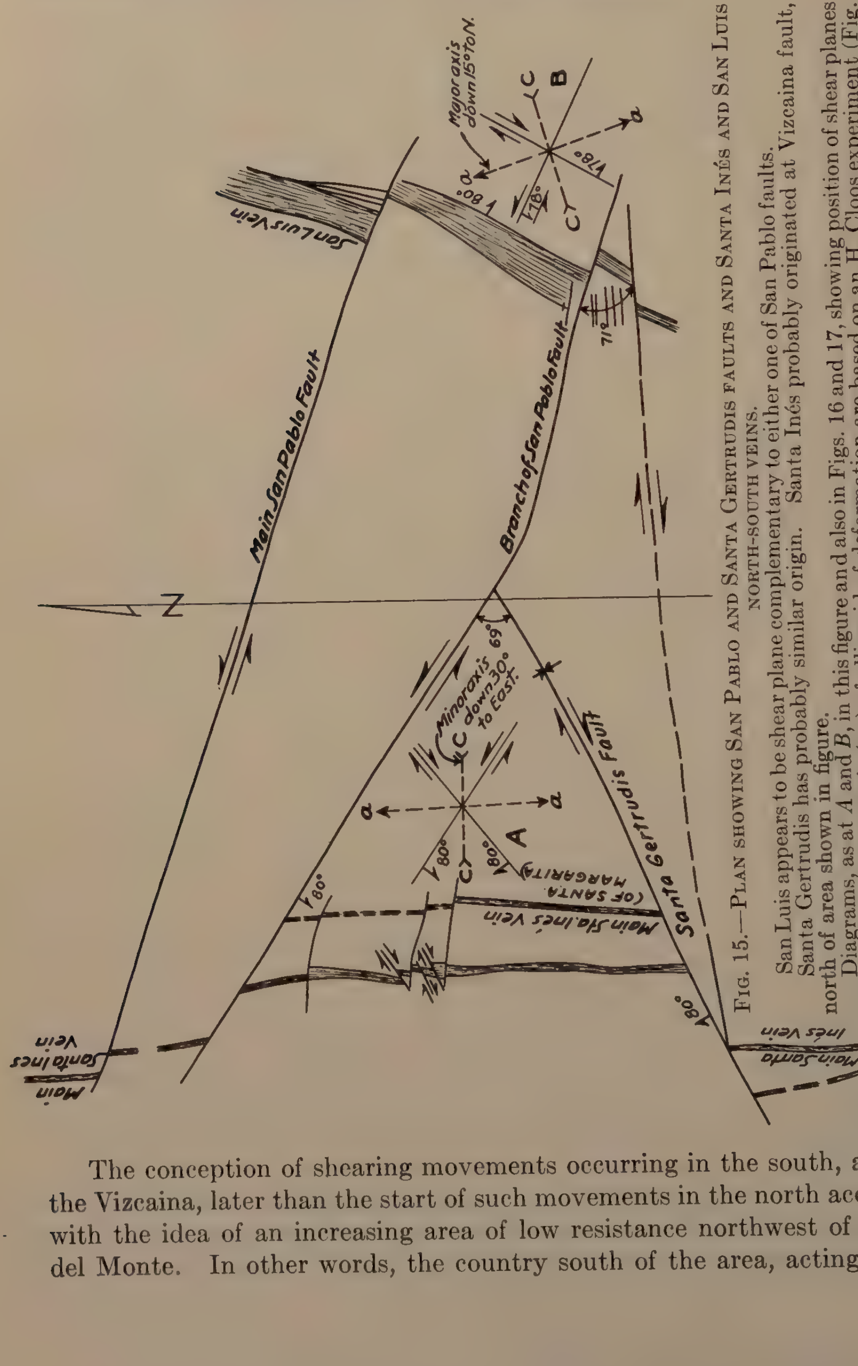


FIG. 15.—PLAN SHOWING SAN PABLO AND SANTA GERTRUDIS FAULTS AND SANTA INÉS AND SAN LUIS NORTH-SOUTH VEINS.

San Luis appears to be shear plane complementary to either one of San Pablo faults. Santa Gertrudis has probably similar origin. Santa Inés probably originated at Vizcaina fault, north of area shown in figure.

Diagrams, as at *A* and *B*, in this figure and also in Figs. 16 and 17, showing position of shear planes and major axis (*a-a*) and minor axis (*c-c*) of ellipsoid of deformation are based on an H. Cloos experiment (Fig. 9). They represent, in some cases, inclined positions for that ellipsoid, corresponding to observed dips for shear planes in field cases compared with the experiment. Fig. 18 shows orientation of tension fissure or "feather joint," with inclined position of ellipsoid.

The conception of shearing movements occurring in the south, along the Vizcaina, later than the start of such movements in the north accords with the idea of an increasing area of low resistance northwest of Real del Monte. In other words, the country south of the area, acting as a

buttress corresponding to the right-hand base block of the experiment, was failing through enlargement to the south of the area of low resistance.

It has been said that horizontal movement on the San Pablo fault probably occurred after such movements on the Vizcaina had practically ceased, and that there was no proof that the San Pablo belonged to the older fracture system exemplified by the Vizcaina. Referring to the experiment of Fig. 11, it may be noted again that continued sinking of the graben finally planed off the salient; when this happened, the antithetic or right-dipping shear planes ceased to form; the synthetic or left-dipping planes continued to develop, and invaded the area formerly closed to them, the neighborhood of the salient. Fig. 6 suggests that the San Pablo, with many parallel faults between it and the Vizcaina, planed off the Vizcaina "salient," and the free slipping, the north side moving west, that had been held up by that salient was resumed. The development of many planes sympathetic with the San Pablo accords with the results of the experiment. Also, the free slipping along the San Pablo was but little conducive to the setting up of strong shearing stresses through the country, hence the San Pablo produced only two important secondary shear planes, now to be described.* Fig. 15 shows, among other things, that the San Luis fracture is oriented with respect to either of two branches of the San Pablo, that it could be a shear plane complementary to either San Pablo branch. (The pulling apart of the San Luis walls by intermineralization horizontal movement on the San Pablo branches is also shown.) Referring to Fig. 6, the anomalous strike of the San Luis, compared with that of the other north-south fractures, is apparent.

Fig. 6 shows another fracture, the Santa Gertrudis fault, which does not differ greatly in attitude, near the San Pablo fault, from the San Luis. Fig. 15 shows that the Santa Gertrudis is probably a shear plane complementary to the south San Pablo branch; it strikes somewhat differently from the San Luis because of the varying strike of its parent fracture, the south San Pablo branch. The fact that the San Pablo branch continues east, while the Santa Gertrudis intersects it and stops, confirms the idea that the San Pablo is the primary fault, the Santa Gertrudis the secondary.

If this is the way in which the Santa Gertrudis fault originated, it must have been more rotated than the San Luis, for over most of its length the Santa Gertrudis strikes southwest; only near the San Pablo

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* The Dios te Guie and San Sabás fractures could, by their orientation, have originated at the San Pablo fault as readily as at the Vizcaina. But the rather large displacements of these fractures by the Vizcaina suggest that they were in existence early in the period of horizontal movement there, and therefore probably before any horizontal movement or shearing stresses had started on the San Pablo—probably before the formation of that fault.

fault does its strike approach that of the San Luis. The horizontal displacement on the Santa Gertrudis, the south side moving west, accords with the direction of rotation demanded; and the sharp change in strike on the San Pablo branch close to the point of intersection with the Santa Gertrudis, could well be the pivot for the rotation, corresponding to the salient in the experiment.

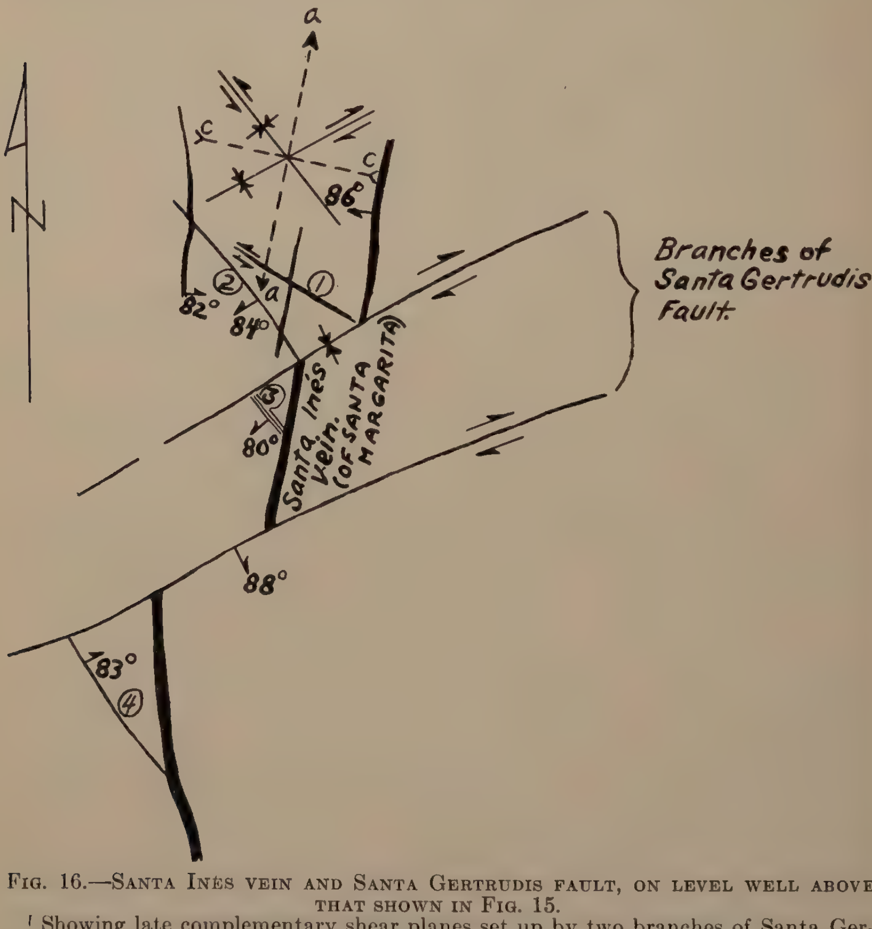

FIG. 16.—SANTA INÉS VEIN AND SANTA GERTRUDIS FAULT, ON LEVEL WELL ABOVE THAT SHOWN IN FIG. 15.

Showing late complementary shear planes set up by two branches of Santa Gertrudis fault. Numbers in circles indicate late complementary shear planes.

A few details are of interest here for the confirmation they give to ideas derived from study of the area as a whole. Fig. 16 shows the Santa Inés vein of Santa Margarita at its intersection with two branches of the Santa Gertrudis fault. The Santa Inés is a shear plane originating from counterclockwise shear; i.e., probably at the Vizcaina fault. Shearing on the Santa Gertrudis was clockwise, hence the Santa Inés was not properly oriented to partake in the movements. Along the Santa Ger-

trudis, however, secondary shear planes were set up, cutting across and sometimes displacing, members of the Santa Inés fracture system. This happened before the close of the period of mineralization, for these northwest shear planes, those complementary to the Santa Gertrudis, were mineralized and in places carry good silver ore. The strong movement

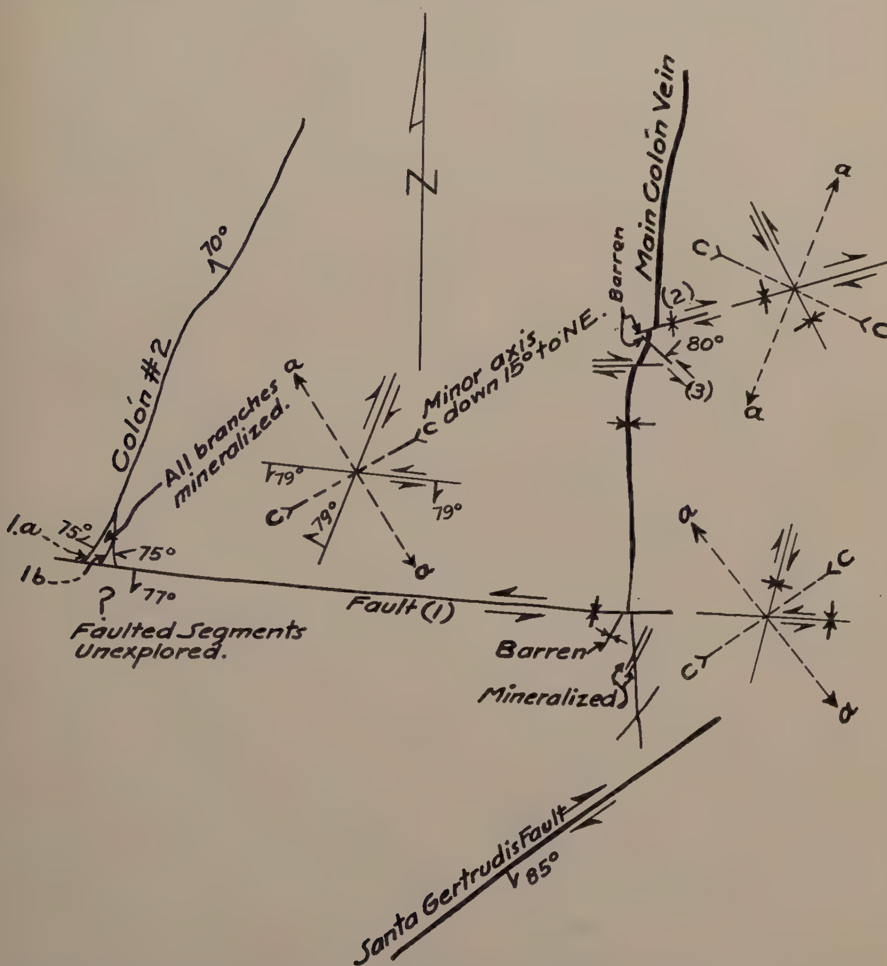

FIG. 17.—COLÓN MAIN VEIN AND ITS BRANCH, COLÓN NO. 2, NORTH OF SANTA GERTRUDIS FAULT.

on the Santa Gertrudis itself prevented ore deposition there, but the Santa Gertrudis carries sparse vein matter in places.

Fig. 17 shows the main Colón vein and its branch, Colón No. 2, just north of the Santa Gertrudis fault. Minor faults, both of the Santa Gertrudis and San Pablo systems, dislocate the main Colón vein. It is evident from the discussion above that the counterclockwise shearing,

as exemplified by movement along the San Pablo and sympathetic faults, had long been active, its area of activity extending from north to south, when the new, and probably sudden, clockwise shear, represented by the displacements on the Santa Gertrudis fault, came into being. The area of Fig. 17 is one wherein the two areas dominated, respectively, by the two shear couples merge. Fault 1 in the figure belongs to the San Pablo family, for it is not oriented in such a way as to be a shear plane comple-

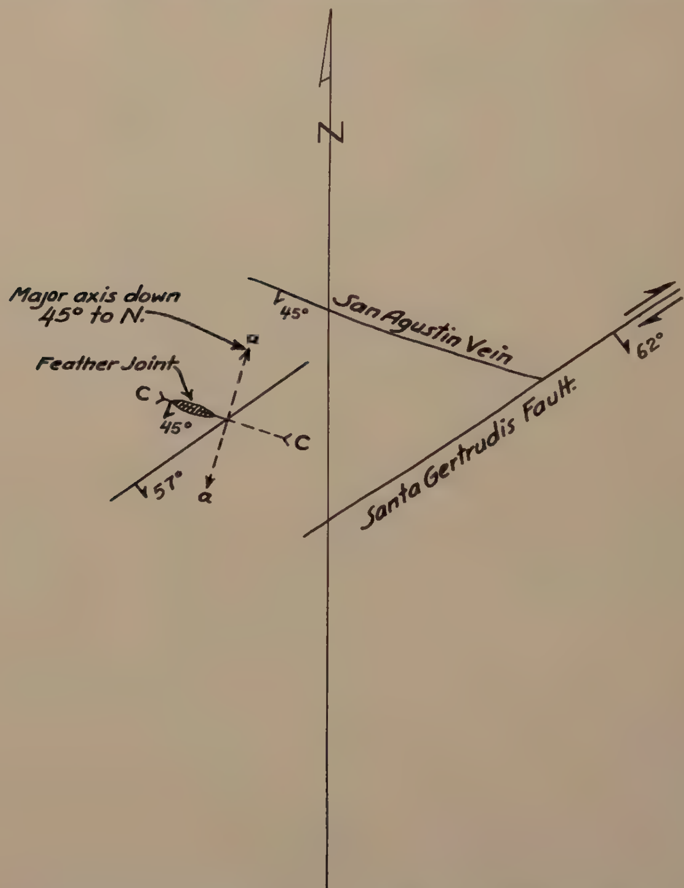

FIG. 18.—SAN AGUSTIN VEIN ORIENTED AS FEATHER JOINT.
Based on experiments by H. and E. Cloos.

mentary to the Santa Gertrudis fault. It is so oriented that the Colón No. 2 vein could have been its complementary shear plane; if not, it acted as such, for it was rotated by the movement north of fault 1, and younger complementary shear planes (vein branches 1*a*, 1*b*) were set up, making the correct initial angle with fault 1. Similar younger complementary planes were set up near the main Colón vein and doubtless elsewhere as well, though exploration is largely confined to following

known veins. Most of these younger complementary planes are mineralized; the counterclockwise shear was still going on while vein matter was being deposited.

Faults 2 and 3, on the other hand, are respectively sympathetic with, and complementary to, the Santa Gertrudis fault. They represent clockwise shear, hence they have no genetic connection with the main Colón vein. They are later than that vein, and are barren. These facts suggest that the clockwise shear was later than the counterclockwise shear, and this, in the main, is thought to be true; but Fig. 16 shows that even the clockwise shear started before the close of the period of mineralization. Naturally, once the strong clockwise shear became active along the Santa Gertrudis fault remnants of the counterclockwise shearing stress would be wiped out.

Returning now to consideration of the clockwise shear along the Santa Gertrudis fault. The Southeast, Ures, No. 1 and San Agustin veins, shown in Fig. 6, are believed to be feather joints[11] formed contemporaneously as a result of tensional stresses set up by the strong shearing, at or close to a stretch of marked irregularity in strike on the Santa Gertrudis fault (Figs. 6 and 18). Their age appears to correspond to the period of horizontal movement on the Santa Gertrudis fault (field evidence); they seem to quit at that fault, and their genesis appears definitely tied up with the Santa Gertrudis fault. They might possibly be secondary shear planes, but their appearance strongly suggests that they are tension or gash fractures. Both secondary shear planes and feather joints appear in nature and in experiments, but seldom or never together. I do not know why this is; the suggestion, to me, is that where the stress overwhelmingly exceeds the strength of the rock, and is applied suddenly, as very possibly in the case at hand, feather joints tend to form; where it is not greatly in excess, and applied perhaps rather gradually, secondary shear planes form.

The Santa Gertrudis fault and its secondary shear planes (Fig. 16) may be compared to the experiment of Fig. 11. To complete the comparison, an area of low resistance is needed, southwest of Real del Monte. This will be discussed, together with the other supposed area of low resistance, northwest of the area.

Accordance of the Suggested Mechanics with the Facts

The preceding discussion of the Real del Monte area has, unavoidably, been long and complex. It is well at this point to review the facts given at the start of description of the area and to see how these facts are explained by the suggested mechanics. The facts, with the comments now made possible, follow.

1. The north-south fractures form a definite belt, show rather even spacing, and are locally discontinuous. The well defined belt was caused

by the localization of the fractures by the stretches of irregularities, opposite each other, on the Gran Compaña and Vizcaina faults. The even spacing results from the fact that the shearing stresses were regional, and the rock fairly homogeneous. The fractures dip steeply because the shearing stresses acted nearly horizontally. They are discontinuous because they are secondary shear planes, some of which originated in one fault block and some in another.

2. The Gran Compaña and Vizcaina show marked irregularities in their fault surfaces, which acted like the salient in the experiment and caused the north-south fractures.

3. Where faults of the N.75°W. system cross the belt of north-south fractures, they show more strong gouge than elsewhere. The gouge was caused by local resistance to the tendency for the ground to move, offered by the irregularities, and by the fact that these irregularities, by holding up the movement, induced differential movement of the fault blocks bounded by other, straighter northwest faults.

4. The N.75°W. and north-south fault systems are mutually displacing, and their habits of displacement are fixed for each set. A comparison with the experiment of Fig. 11 is illuminating.

5. South of the Vizcaina the N.75°W. system is poorly developed, but a strong northeast fault, the Santa Gertrudis, appears there. It displaces the north-south fractures always with the south side moving west. Movement westward, owing to the stress from the east indicated by the minor axis of the ellipsoid shown in Fig. 12, was comparatively easy on most of the pre-existing N.75°W. faults north of the Vizcaina. But the crooked Vizcaina blocked the general movement, especially since it had no parallel faults south of it. Hence a new parallel fault had to be set up to plane off the irregularities on the Vizcaina; i.e., the San Pablo. This took force, and at the start of the formation of the San Pablo, before the fault had been completed from east to west, strong shearing stresses were created in the country rock. These formed the San Luis vein and the Santa Gertrudis fault. Now the Santa Gertrudis was so placed that it was easier for the westward-moving ground to turn and glide southwest along the south side of the newly formed Santa Gertrudis than to continue west along the north side of the San Pablo (Fig. 15). This was especially true because, as will be shown below, an area of low resistance probably lay southwest of the Santa Gertrudis fault. This combination of facts accounts for the very strong movement on the Santa Gertrudis, and for the direction of the movement.

6. The movements on the N.75°W., north-south, and northeast faults were horizontal. The mechanics called on require this.

7. Most of the N.75°W. faults are older than the north-south fractures. The mechanics do not require this; but the movements required by the mechanics were much easier because of this fact.

8. The north-south fractures were forming at the time of the horizontal movements on the N.75°W. faults. Obviously this is required by the mechanics of the experiment of Fig. 11. It forms one of the principal points of the analogy.

9. The San Pablo and Santa Gertrudis horizontal movements were later than those on the Vizcaina. This is a necessary feature of the theory of formation of those faults, just outlined.

10. The veins of northern Real del Monte show largely the earliest vein minerals. This follows from the ideas of the start of shearing stresses outlined above.

Relation of Late Fracturing at Real del Monte to Geologic History of Sierra de Pachuca

The supposed immediate causes for the late fracturing of the Real del Monte area have been given. What more fundamental causes lay behind them? On the answer to this question depends any real understanding of the area.

There follows, for convenience, a "time table" of the geologic events believed to have taken place in the Sierra de Pachuca during the epoch of most intense intrusion, extrusion and diastrophism.

1. Extrusion of the pre-rhyolite-flow volcanics; i.e., the augite-andesite series. This preceded the epoch of disturbance by an unknown period; but it is doubtful whether the period was long, since, according to Thayer, all the volcanics are of Miocene age.

2. Beginning of a fracture system striking about N.75°W. in the Pachuca district, but more nearly east-west in the Sierra as a whole.

3. Intrusion of quartz-rich dacite, accompanied by uplift and warping of the augite-andesite series. Stocks and dikes intruded along fractures of the N.75°W. system.

4. Extrusion of rhyolite flows, the lava ascending through necklike channels located within the two areas of the flows (Fig. 2).

5. Extrusion of normal dacite flows, the lava ascending through channels practically identical with the channels utilized by the rhyolite. Intrusion of a number of normal dacite dikes, chiefly along fractures of the N.75°W. system, which was still developing.

6. Completion of the dip-faulting (synthetic movements) on the fractures of the N.75°W. system. Vertical shoving by the magma, or the possible large magmatic movements that permitted free dropping of fault blocks, had ceased. The main portions of the assumed magma had nearly or quite consolidated.

7. Intrusion of the quartz-porphyry dikes; second period of rhyolite intrusion. N.75°W. system attained its greatest development; it was still being squeezed open by the east-west compression, according to the theory outlined.

8. Just before the close of the period of intrusion of the quartz-porphyry dikes, horizontal movements started on certain faults of the N.75°W. system in the Real del Monte area; north-south fractures formed there; the Santa Gertrudis fault, allied in origin to the north-south fractures, was set up; the mineralization took place.

From stages 2 to 8, and particularly from stages 4 to 8, the history forms a closely linked sequence of events compressed into a relatively short period. Some of the events undoubtedly overlapped preceding or succeeding events. The start of horizontal movements, therefore, came at a time very closely following the extrusion of the rhyolite and dacite flows. To show how close the time relations were, the fact may be noted that the upper horizons of the dacites probably are younger than some of the quartz-porphyry dikes.

Field study in a number of places shows that both rhyolite and dacite intrusive masses are much smaller at shallow depths than at the present surface; among these intrusions are some thought to represent vertical channels for rhyolite or dacite extrusions. This means that the great mass of flows of each type was essentially unreplaced in the chambers from which it rose. Both the rhyolite and dacite flows spread out over the country for considerable distances away from the highly localized areas of volcanoes that gave them origin. Hence these volcanic centers became localities of low rock density; i.e., of low resistance to stress. The spreading out of the flows away from these centers would greatly lessen the tendency toward strong slumping (due to weight accumulated at and near the surface above the volcanic channels) that would have occurred had not the flows spread widely; such slumping would have partly refilled the semivoids beneath.

These localities of low resistance formed at the time of the extrusion of the rhyolite and dacite flows, and this time, as has been shown, was very shortly before the beginning of the horizontal movements and the formation of the north-south fractures in Real del Monte.

In the lower part of Fig. 10*b*, where some slipping occurs along the major, left-dipping, synthetic shear plane, but where some antithetic faults are nevertheless set up, the actual direction of movement of particles of the clay, directed toward the area of low resistance in the experiment, is practically parallel to the major shear plane. The slight divergence from parallelism is due, of course, to slight rotation of the active block. This part of Fig. 10*b* corresponds to the northern Real del Monte area, and the direction of movement for that area, very nearly identical with the direction of the active member of the shear couple there, has been shown on Fig. 2. The direction of movement for the clockwise shear along the Santa Gertrudis fault is also shown.

The direction of movement for northern Real del Monte points straight at the northwest volcanic center; that for southern Real del

Monte points between the major volcanic center there and a group of minor centers, though the location of the major volcanic center may not be accurate because of poor and confusing exposures. The analogy with the experiment of Fig. 11 is now complete.

The flow planes in three rhyolite plugs have been partly mapped, and their generalized attitudes are shown in Fig. 2. The two smaller plugs gave origin to early rhyolite extrusions; the age of the third and largest may correspond with that of the other two, or may belong with the second rhyolitic period. The flow planes, away from the contact, have a constant trend, regional in character. Robert Balk[4] shows that where flow planes are developed the shortest axis of the ellipsoid—the axis of compression—stands perpendicular to the planes. Further, he also shows[3] that where igneous intrusives solidify under regional compression the flow planes (containing the flow lines arranged in the direction of stretching) stand perpendicular to the compression, and have a regional trend, not parallel to the contacts except where close to them. The flow planes of Fig. 2 show that the plugs solidified under regional compression acting between east-west and northeast-southwest. As has been stated in the paragraph under Structure and History of the Sierra, this direction is intermediate between that followed by the compression supposed to have formed the cross joints and that concerned in the sharp folding along the regional axis northwest of Pachuca. This accords with the notion that the compression acted at first in an east-west direction and then swung around to the northeast. The later direction would have kept the cross joints squeezed open for the admission of successively younger dike types almost as well as the original direction supposedly involved in the formation of the joints.

Here, then, is direct evidence, independent of comparisons with experiments, that a compressive stress, acting about east-west, was active during the intrusion and extrusion of the rhyolite; i.e., at a time close to that of the late fracturing in Real del Monte.

A further confirmation of the ideas set forth from the Pachuca area is that a set of east-west fractures exists there that includes some of the main veins of that area, such as the Tajo, Analcos and other veins shown in Fig. 2. These fractures are demonstrably later than the N.75°W. fractures: they are also later than the quartz-porphyry dikes. While they strike east-west, the fractures form a belt trending northwest. Fig. 19 is a vertical section showing some of these fractures. Obviously they are antithetic faults. The belt they form tends to connect the two volcanic centers. Antithetic faulting requires the presence of lateral "elbow room" to permit the rotation of the fault blocks essential to that type of faulting. It does not require either a vertical force or the ability of the blocks to move strongly down the dips of the faults. At the time of formation of these fractures, the magma had ceased differentiating,

except for the production of vein matter; hence it had probably solidified, at least in its upper portions. With the cessation of vertical magmatic shoving, the era of synthetic faulting would come to a close; with the formation of the two areas of low resistance and low rock density, a direction connecting the two areas of low resistance, along which lateral crustal yielding was possible, would appear. But the horizontal (east-west) regional compressive stress continued past the time of the close of mineralization, as shown among other evidence by horizontal striae on vein quartz, and therefore was active during the formation of these antithetic faults. The ground was stretched strongly north-south, between the two volcanic centers. This horizontal crustal yielding permitted the antithetic faulting, and may have formed the faults themselves, just as the flat-dipping transverse antithetic faults (*Streckflächen*) of Balk were formed by the stretching of the granite massive (ref. 4, Fig. 7).

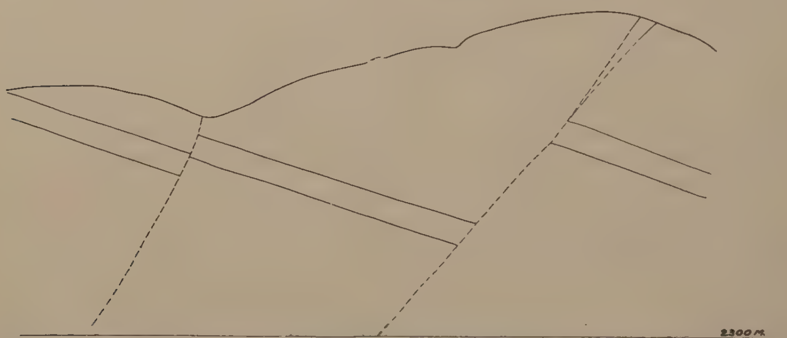

FIG. 19.—VERTICAL SECTION SHOWING PART OF ANTITHETIC FAULT SYSTEM OF PACHUCA AREA, LOOKING WEST.

Summary of the Mechanics

Before the formation of the two localities of low resistance to stress, the country was more or less homogeneous, and reacted to the compressive stress as a unit. During this time the supposed cross joints were formed. But with the start of formation of the two centers of weakness, the simple compressive stress was resolved into two shearing stresses. These operated, respectively, counterclockwise around the north side of the "buttress" formed by the firm ground lying east of a line connecting the two weakened localities, and clockwise around the south side of that buttress (Fig. 2). The shear couples consisted of the force shoving the ground toward the locality of low resistance forming one member, and of the resistance of the buttress forming the other member. The buttress was, of course, ill defined; actually, the "resistance of the buttress" was the sum of the resistances of the irregularities on the N.75°W. faults, for the counterclockwise shear, and was the resistance of the northwest wall of the Santa Gertrudis fault, for the clockwise shear.

These shearing stresses set up the north-south fractures and the other late fractures of Real del Monte.

At the same time, the simple compressive stress was transmitted through the center of the buttress into the Pachuca area. It stretched the ground there in a direction roughly at right angles to itself; the direction of stretching was slightly modified by the positions of the two areas of low resistance, so that the ground was stretched along a line connecting these areas. This caused the Pachuca antithetic faulting.

Progressive enlargement of the two weak localities, called for by comparison with the experiment, might naturally accompany the continued passing upward of lava through these conduits, once the initial channels had been formed.

Relation of Mineralization to Mechanics

Over a great part of Mexico the intrusion and extrusion of rhyolite closely preceded the mineralization. This is a habit of magmatic differentiation in the central Mexican province. At Pachuca, the extrusion of the rhyolites (and dacites), combined with the location of the two volcanic centers and the long-operating east-west compressive stress, caused the formation of the north-south fractures in Real del Monte and the antithetic fractures in Pachuca. It is therefore hardly a coincidence that the vein matter came in as these fractures were forming.

Significance of the Paper

The field work in the Pachuca district has been carefully done; and if the ideas derived from that work and from recent conceptions of the mechanics of intrusion, as presented in this paper, are well founded they would seem to have many possible applications elsewhere. The work of H. Cloos and others has shown that intrusives are remarkably sensitive reflectors of tectonic stress operating at the time of their emplacement. Where intrusives are closely linked in time with ore-bearing fractures, knowledge of stresses active at the time of the intrusion may be of the utmost aid to the mining geologist. A conception of the fracture types set up in consolidating intrusive masses is important, not only in connection with mining camps situated within such a mass, but in connection with the probably far more numerous mining districts of which the veins are in the covers lying above buried intrusives of large size.

The following up of these ideas implies the careful mapping of all features connected with intrusives, such as flow planes and flow lines, together with marginal fissures and the other types of faults.

The desirability of much more extensive experimentation along the pioneering lines followed by Hans and Ernst Cloos is very plain. Such experiments should be conducted by workers with many different kinds

of field experience, so that they may attempt to duplicate in the laboratory occurrences in nature that they have seen in the field.

Acknowledgments

I am indebted to Mr. George Scarfe and Mr. C. D. Hulin, my predecessors at Pachuca, not only for many of the fundamental data upon which the ideas of this paper are based, but also for valuable generalizations throwing light on the problems involved. I also owe much to Dr. Ernest Cloos for painstaking criticism of this paper and for other aid, and to Dr. Robert Balk for furnishing copies of a number of his significant papers; also to Prof. J. T. Singewald, Jr., Prof. C. H. Behre, Jr., and Mr. M. D. Harbaugh for constructive comments on the manuscript. Finally, thanks are due Messrs. C. F. Moore and M. H. Kuryla of the United States Smelting, Refining and Mining Co. for permission to publish the paper.

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DISCUSSION

(*Edward Sampson presiding*)

J. B. Platts,* Wallace, Idaho (written discussion).—Mr. Wisser's observations and conclusions agree closely with those of the writer. Antithetic faulting has been an important factor in the formation of commercial ore deposits, as may be shown by numerous examples in other districts. Conditions differ in different districts, but always there must be fractures of some kind that tend to be pulled open by the fault movements and so form channels that may be traversed for long periods by ore-

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* Mining Geologist.

making solutions. Antithetic faults in hard rock serve the purpose best. Synthetic faults, whether normal or thrust, tend to tighten with the movement and to be plugged with gouge and they seldom form suitable channels for the long-continued solution movements that are necessary to complete the great amount of replacement and deposition in a large orebody.

There are striking parallels between geologic conditions in the Pachuca district, as described by Mr. Wisser, and the conditions found in the Coeur d'Alene district. As at Pachuca, the entire region has been subjected to a long-continued side thrust or pressure and, although the thrust happens to have been in nearly the opposite direction (from the northwest instead of from the east) the general effect is the same—a series of northwest and southeast-striking fault fissures over a wide area and forming a shear zone on a grand scale. Cross fractures striking northeast and east have been rotated clockwise with the fault blocks until many of them strike east and southeast. Examples of these cross fractures are to be seen in the workings of the Bunker Hill and Sullivan mine and elsewhere in the Wardner area, where much of the ore is associated with northeast-striking cross veins. An intrusive mass of granitic rock is understood to underlie this part of the district, although it does not appear at the surface. Farther east, in the so-called dry belt, clockwise rotation has been more pronounced. There the system of vein fissures splits into many branches, some of them trending toward the southeast nearly parallel with the major faults while others trend a little north of east. It is noteworthy that these northeast fractures are more likely to contain ore than fractures striking parallel to the major fault movements. Some of the development work being carried on in that region has failed to find ore because the operators have overlooked this point. Because of antithetic faulting the northeast fractures have tended to remain open or permeable. There are a few apparent exceptions.

In the Burke and Mullan sections of the district, conditions are more complex. There the northwest-southeast fault system is superimposed on an older north-south system. The "Gem granite" (quartz monzonite) intruded this north-south shear zone and has since been exposed at the surface by erosion. It is probable that the mineralization of that part of the district was started in the old system of fissures, the metals being deposited in fractures striking northeast and east, across the north-south system. The north-south shear couples rotated the fault blocks to the left or counterclockwise, and the later and longer continued thrust from the northwest rotated them back in the opposite direction and continued until most of them now strike east-west and southeast. There are also some exceptions here and not all of the orebodies were deposited in fractures that may be attributed directly to antithetic faulting, yet it is evident that this kind of faulting had a great deal to do with preparing the ground for the ore deposits.

E. Wisser (written discussion).—Mr. Platts' comparison between the Coeur d'Alene district and Pachuca is of great interest to me. Several Mexican districts appeared to offer similarities to Pachuca, but I never had the opportunity to study them in sufficient detail to be sure. Clearly Mr. Platts has reached his conclusions regarding the tectonics of the Coeur d'Alene district quite independently of any work of mine at Pachuca. I think there are few of us engaged in attempting to solve the riddle of ore localization that feel the ground any too firm under our feet. It is a comfort to find confirmation of my ideas in a district entirely unknown to me, and hence never the pastureland for any hobbyhorse of mine.

Mr. Platts' generalization contrasting tight, gougy synthetic faults, the loci of strong shearing movement, with antithetic faults in competent rock, tending to remain open to receive the mineralizing solutions, is sound and fundamental. It is a logical step ahead of the ideas I have sought to develop in my paper, and I am indebted to him for expressing this idea in stimulating form.

Geology of Lead-zinc-copper Deposits at Buchans, Newfoundland

By P. W. George,* Member A.I.M.E.

(New York Meeting, February, 1937)

This paper presents geological data regarding deposits of over 7,500,000 tons of fine-grained sulphide ore in barite gangue. A series of pyroclastics and arkoses was intruded by sills of quartz porphyry in connection with batholithic intrusions of granite 3 miles from Buchans. Gentle folding, with fracturing and shearing of tuff beds and intruded quartz porphyry, accompanied and followed the intrusion of granite and porphyry. The sulphide-barite mineralization is localized primarily in fractured and sheared tuff beds, but also in the sheared quartz porphyry. Two structures have proved particularly favorable for mineralization—one, the center of an anticline; the other, a series of weak, incompetent tuff beds intruded by thin sills of quartz porphyry and located between strong beds of arkose. During the folding the weak tuff with intruded quartz porphyry was intensely sheared and contorted and subsequently mineralized by sulphides and barite.

Introduction

Buchans is 5 miles north of Red Indian Lake, in Central Newfoundland, at an elevation of 900 ft. above sea level and 400 ft. above Red Indian Lake, and is reached by a 37-mile private railway from Millertown Junction.

The first ore body found at Buchans was discovered some 30 years ago in the bed of Buchans River, by Matty Mitchell, an Indian woodsman employed by the Anglo-Newfoundland Development Co. Ltd. In the years following the discovery the ore was developed by shaft-sinking and drifting, with results indicating a lenticular ore body in a shear zone, containing about 100,000 tons of extremely fine-grained sulphide ore in barite gangue averaging about 0.05 oz. Au, 4 oz. Ag, 1.5 per cent Cu, 10 per cent Pb, 18 per cent Zn. Because a profitable metallurgical treatment for the ore was not found, the mine was closed down in 1911.

In 1915, Mr. H. A. Guess, Vice President of the American Smelting & Refining Co., learned of the Buchans River ore body, and after metallurgical tests had been carried on from time to time in his company's testing plant a satisfactory separation of the sulphides by selective flotation was finally effected in 1925. Prospecting for additional ore bodies was begun in 1926, utilizing geophysical methods under the direction of Hans Lund-

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* Manager, Buchans Mining Company, Ltd., Buchans, Newfoundland.

berg[1]. By this work and by diamond drilling, large ore bodies of a type similar to the original discovery at Buchans River were found in two new locations. The larger of the two, called "Lucky Strike," was encountered some 3500 ft. west of Buchans River mine, and the other, called "Oriental," 2000 ft. east of the original discovery. The aggregate of these newly discovered ore bodies, after development by diamond drilling and underground work, represented about 6,600,000 tons averaging approximately as follows: Au, 0.05 oz.; Ag, 3.1 oz.; Cu, 1.4 per cent; Pb, 8.5 per cent; Zn, 17.4 per cent; Fe, 7.6 per cent; $BaSO_4$, 30.0 per cent.

Milling commenced in the fall of 1928 and was carried on at a rate of about 500 tons per day until July 1931, when the capacity of the mill was increased to 1200 tons per day.

During the first years of Buchans development after 1926, geological work was mainly carried on by Hans Lundberg and W. H. Newhouse. Dr. Newhouse published a paper on the geology and ore deposits of Buchans, based on work done during the summers of 1927, 1928 and 1929, and including interesting microscopical data[2]. Since Dr. Newhouse's paper was published, abundant new geological data have accumulated through mine development, diamond drilling and further study of the ore deposits and rock outcrops in the adjoining areas by members of the regular mine staff, assisted in the summers of 1930 to 1933 by Hans Lundberg. Of the mine staff, particular credit is due G. G. Gilchrist, Mine Superintendent, for valuable geological observations, and G. W. Moore, Chief Engineer, for careful recording of diamond-drill cores and mapping of the geology. In this paper, it is the writer's aim to describe the ore deposits and the geology of Buchans from the viewpoint of the additional information now available, assembled by the writer and augmented by data from his personal observations and study during the past seven years.

Topography

The country near Buchans is flat or gently sloping. It consists mostly of a series of bogs dotted with lakes, and separated by low rounded ridges and hills covered with dense growth of small spruce and fir, except where granite is exposed. Rock outcrops are few and generally of small extent, except in beds of streams, as most of the area is covered by glacial drift averaging from 10 to 50 ft. in depth. North and west of Buchans a few hills rise to elevation of 1500 to 1900 ft. above sea level.

Many of the ridges, bogs and upper parts of streams, and the general strike of the gently folded rocks parallel Red Indian Lake. The lower

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[1] H. Lundberg: Recent Results in Electrical Prospecting for Ore. *Trans.* A.I.M.E (1929) **81,** 118–122, Geophysical Prospecting.

[2] W. H. Newhouse: Geology and Ore Deposits of Buchans, Newfoundland *Econ. Geol.* (1931) **26,** 399–414.

FIG. 1.—GEOLOGICAL MAP OF AREA AROUND BUCHANS. SHOWING STRUCTURAL RELATIONSHIP OF BUCHANS SERIES TO YOUNGER INTRUSIVES.

parts of the streams run transverse to the structure (Fig. 1). The drainage pattern and the configuration of bogs, ridges and hills generally conform with the direction of movements of ice, as recorded by glacial striae, *roches moutonnées* and boulder fans. Mineralized boulders from Lucky Strike have been transported westwards by the earlier ice sheet, which moved in southwesterly direction towards the coast. The last glacial movement, which left fewer marks on the landscape than the earlier, moved from the neighborhood of Hinds Lake towards Buchans, or in direction of about S. 20° E. East of Red Indian Lake the ice moved in a northeasterly direction.

The Buchans Series

The oldest rocks in the Red Indian Lake region form a series consisting mainly of basic lavas, agglomerates, tuffs and arkose believed to be of Ordovician age[2,3] and termed "Buchans Series" by W. H. Newhouse. In the vicinity of the mines the main flows of basic lava are from 100 to 1000 ft. thick. Between the flows are beds of arkose, agglomerate, tuffs and other fine-grained siliceous sediments, in places aggregating a total thickness of over 1000 ft.

Lava Flows.—The lava flows near the ore deposits are andesitic or basaltic and mostly amygdaloidal. Amygdules and fractures in the lava are generally filled with calcite; sometimes with chlorite and quartz. Some of the lava flows are colored red by secondary hematite. This is particularly noticeable in a lava flow south of Oriental (Fig. 2) and in the upper part of the lava cut by the 350-ft. level west of Oriental (Fig. 3). Both these occurrences are probably parts of the same flow.

Tuff and Agglomerate.—Tuff and agglomerate of irregular extent, and in thickness ranging from a few feet to some 800 ft., are generally found above the lava flows, or interbedded with lava, and with arkose and other siliceous sediments. The tuff and agglomerate are silicified and vary in color from light gray to green. Banding is plainly visible in parts of the silicified tuff.

Arkose and Conglomerate.—Arkose, consisting mostly of quartz and feldspar grains with a few larger fragments of granite and greenstone, is found interbedded with lava and pyroclastics in some areas. At the Oriental mine the arkose beds reach a total thickness of up to 400 ft., although the upper beds have been eroded. Conglomerates of small extent with pebbles of granite and greenstone occur at Old Buchans River mine and at Lucky Strike mine.

Intrusives in Vicinity of Mines

Buchans Series has been intruded by granite, diabase and quartz porphyry of pre-Carboniferous age. Granite and diabase cut Buchans

[3] G. R. Heyl: Geology and Mineral Deposits of the Bay of Exploits Area. Newfoundland Dept. of Natural Resources, Geol. Sec. *Bull.* No. 3 (1936) 11, 12.

FIG. 2*a*.—GEOLOGICAL PLAN OF AREA AROUND BUCHANS, WEST PART.

FIG. 2b.—GEOLOGICAL PLAN OF AREA AROUND BUCHANS EAST PART.

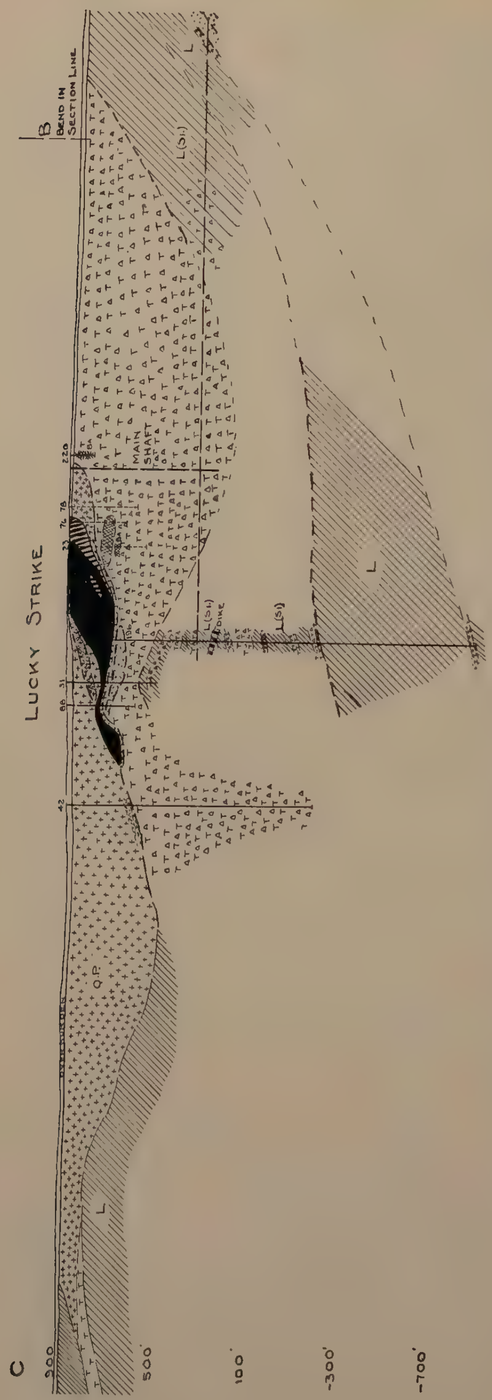


FIG. 3*a*.—GEOLOGICAL SECTION *C-B-A* ALONG LUCKY STRIKE-ORIENTAL MAIN FOLDING AXIS, WEST PART.

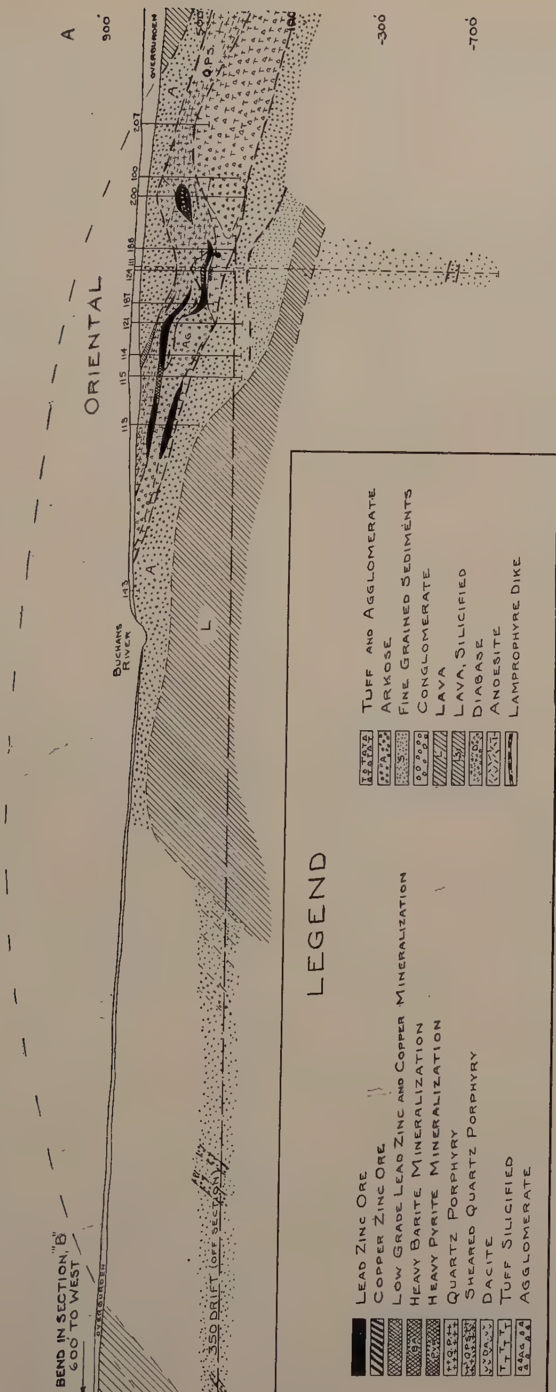


FIG. 3*b*.—GEOLOGICAL SECTION *C-B-A* ALONG LUCKY STRIKE-ORIENTAL MAIN FOLDING AXIS, EAST PART.

Series 3 to 4 miles north, east and west of the mines. These intrusives are parts of large batholiths extending for some 50 miles or more in northeast-southwesterly direction. Sills and bodies of quartz porphyry, dacite and andesite have intruded the sediments and lavas near the

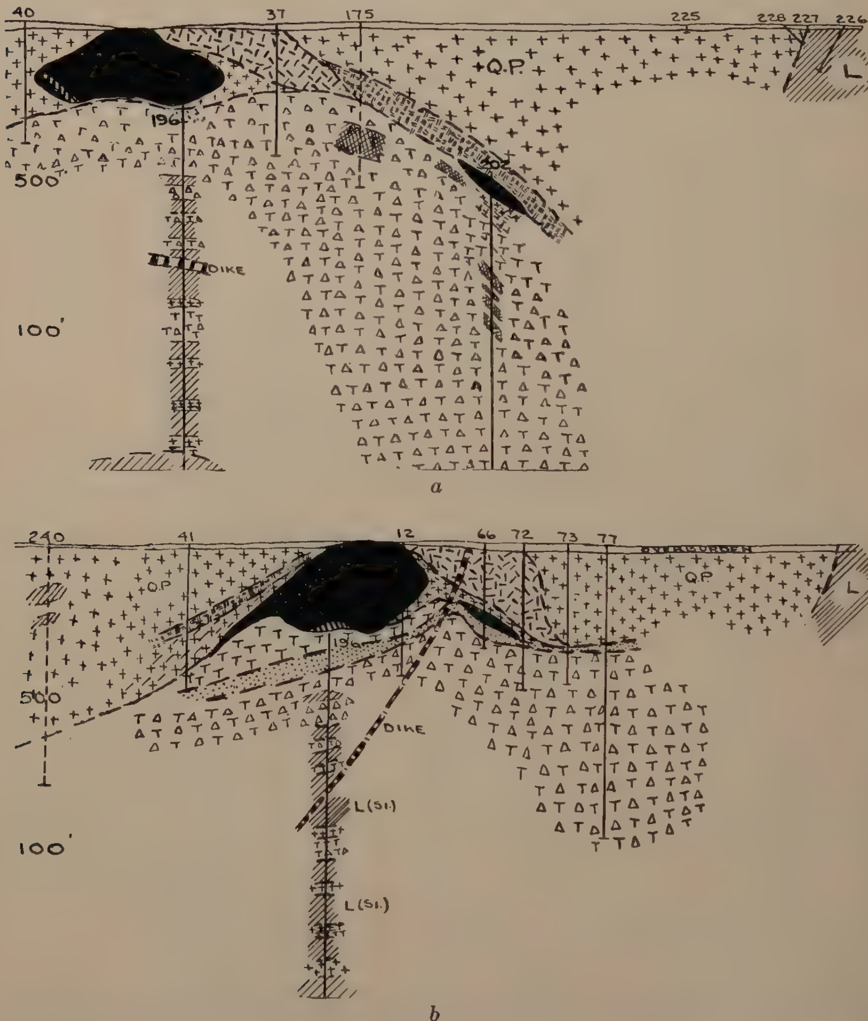


FIG. 4.—GEOLOGICAL SECTIONS, LUCKY STRIKE ORE BODY, LOOKING WEST. *a*, SECTION 7–8; *b*, SECTION 9–10.

mines. Folding and shearing or fracturing of tuff, agglomerate and arkose accompanied and followed the intrusion of the granite, quartz porphyry and dacite. Sulphide and barite mineralization took place after the quartz-porphyry intrusion. Subsequent diabase and lamprophyre dikes cut the pyroclastics, porphyry and ore (Fig. 3–6).

Andesite.—In the hanging wall of the Lucky Strike ore bodies, which mostly consists of quartz porphyry, is a thick sill-like body of altered nonamygdaloidal greenstone, classed as andesite by W. H. Newhouse

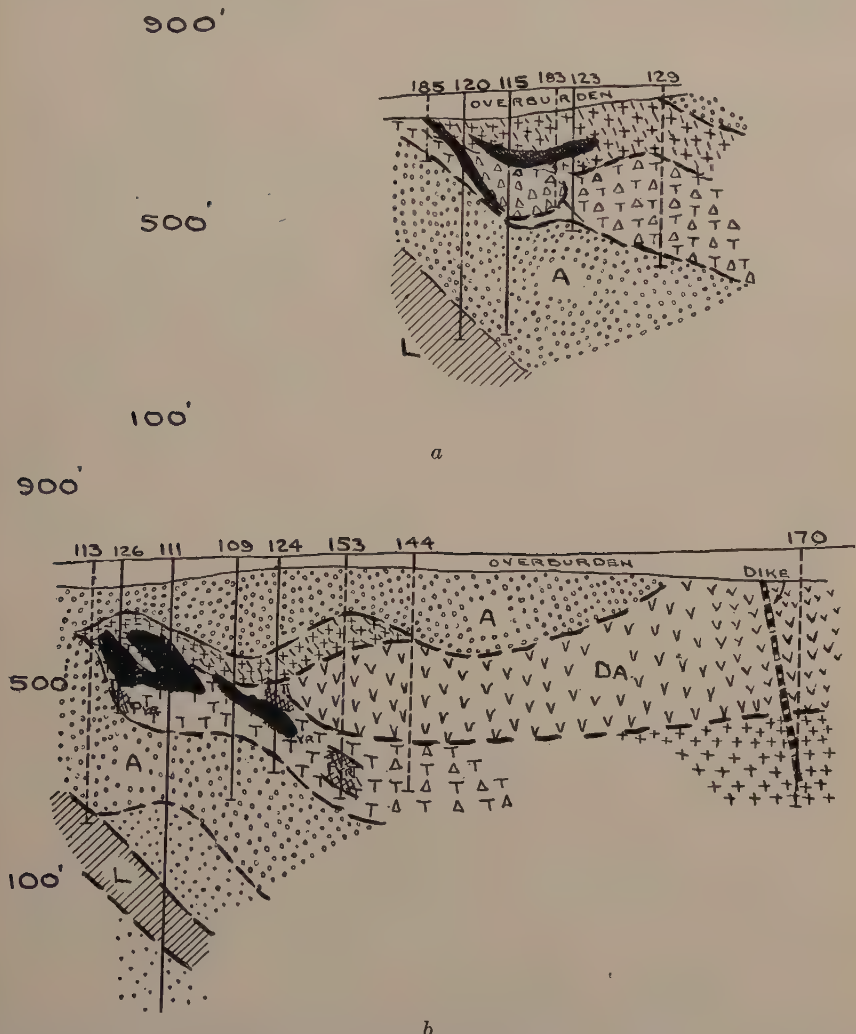
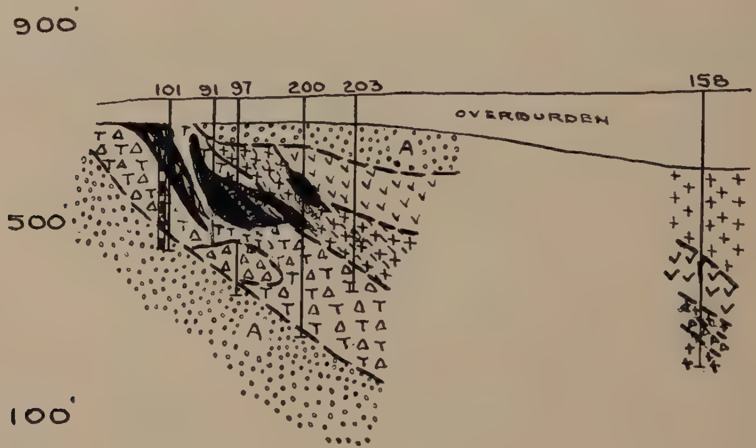


FIG. 5.—GEOLOGICAL SECTIONS, ORIENTAL ORE BODY, LOOKING WEST. *a*, SECTION 13–14; *b*, SECTION 15–16.

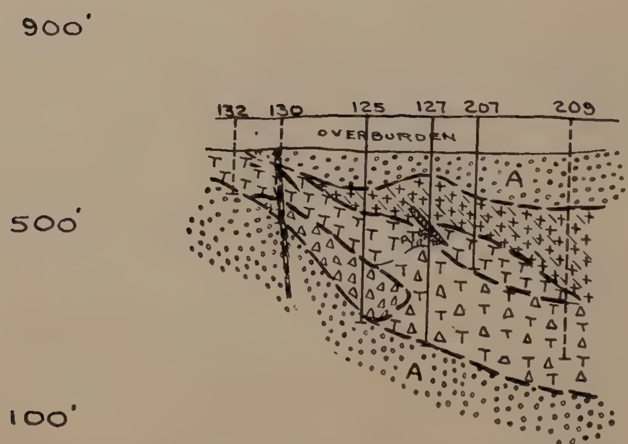
(Fig. 6). The andesite is older than the quartz porphyry, as the latter has absorbed fragments of andesite.

Quartz Porphyry and Dacite.—Of these rocks which have intruded the pyroclastics and lavas as sills, or sill-like masses, varying in thickness from a few inches to 1000 ft., quartz porphyry is by far the most common. It is generally found in the hanging wall of the ore bodies. At Lucky Strike

mine some steeply dipping dikes or tongues of quartz porphyry, striking parallel to the axis of the folds, extend from a thick quartz-porphyry sill into the ore bodies. The dacite at Lucky Strike is apparently older than the quartz porphyry, although some dacite succeeded the quartz porphyry at Oriental mine.



c



d

Fig. 5.—Geological sections, Oriental ore body, looking west. *c*, section 17–18; *d*, section 19–20.

Diabase and Lamprophyre Dikes.—Steeply dipping diabase dikes, ranging in width from about 30 to 100 ft., cut the quartz porphyry, pyroclastics and lava south of Lucky Strike and Oriental. The strike of the diabase closest to Lucky Strike mine is northwest-southeast. About two miles south of Oriental a 30 to 80-ft. wide diabase dike strikes N. 60° E. Several lamprophyre dikes, from a few inches up to 20 ft. in

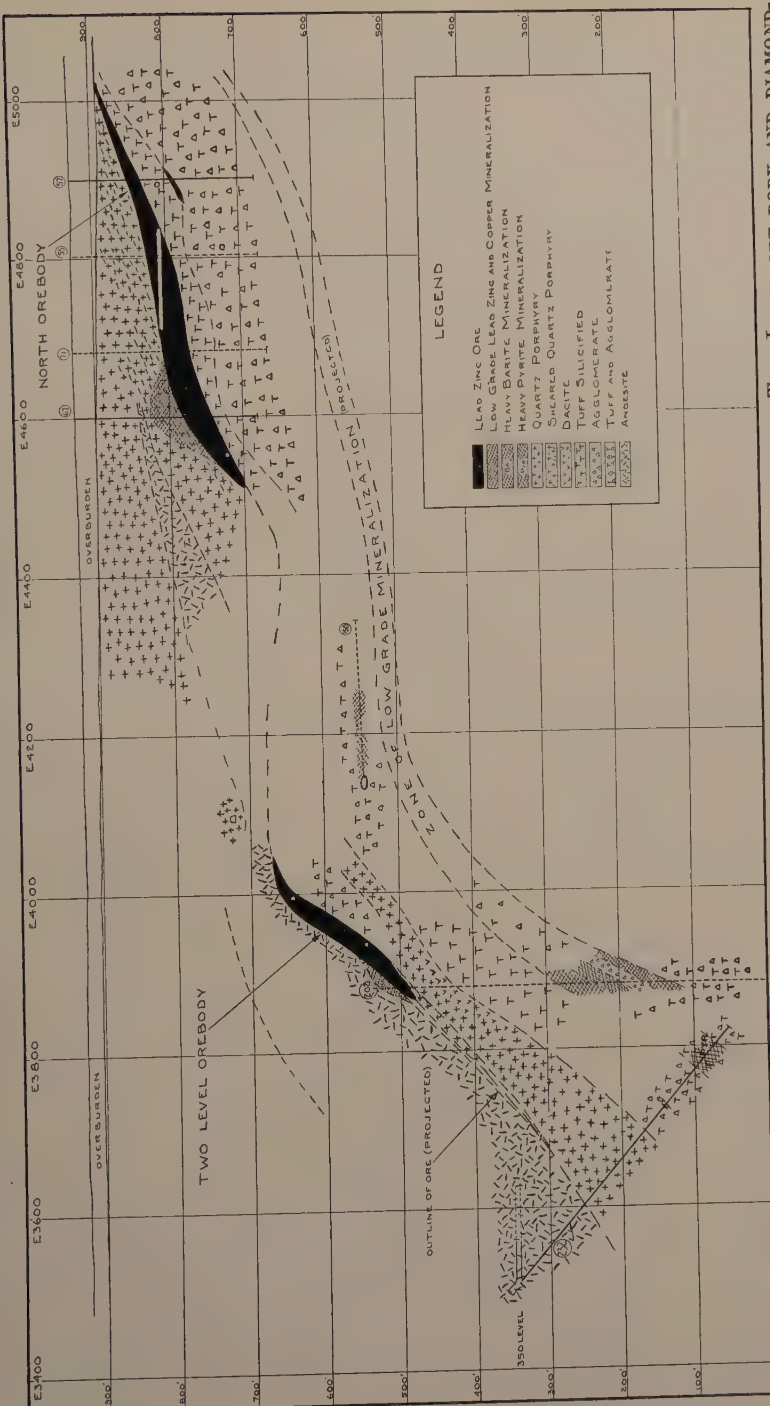


Fig. 6.—Geological section N. 68 W.-S. 68 E. showing Lucky Strike north orebody, Two Level ore body and diamond-drill hole No. 232.

thickness, cut the pyroclastics, quartz porphyry and ore. The strike of these dikes is mostly east-west, parallel to the axis of the folds, with dip about 80° to the north.

Mineralization of Buchans Series and Later Intrusives

The tuff is the principal host rock for the barite-sulphide mineralization. The agglomerate, which generally is andesitic, is in parts chloritized and pyritized and carries minor amounts of relatively coarse-grained lead, zinc and copper sulphides. The lava flows and the arkose do not contain appreciable amounts of sulphides even at contacts with quartz porphyry. The matrix of the two small patches of conglomerate mentioned has been replaced with barite and sulphides to the extent that the conglomerate forms a medium grade lead-zinc ore. Of the intrusives, the quartz-porphyry sills and dikes near the ore bodies are extensively sheared, hydrothermally altered and in parts mineralized with lead and zinc sulphides, barite and pyrite. The andesite and the dacite do not generally contain any sulphides, although the latter rock in some parts is much sheared and altered. Fractures in the lava, in the arkose and in the ore bodies are generally filled with calcite. In the andesite the fractures are cemented with calcite and quartz; in the porphyries with quartz. The tuffs and agglomerates are silicified, but contain also fractures filled with calcite or barite.

For petrographic description of the rocks at Buchans see W. H. Newhouse's paper cited in footnote 2. No other petrographic data are available.

The stratigraphic relations of the lavas, pyroclastics, arkose and intruded porphyry to the sulphide-barite mineralization at Buchans are indicated in the stratigraphic section and in Figs. 2–6.

Structure

The average strike of Buchans series in the Red Indian Lake region is S. 55° W., approximately the same as the general direction of the granite intrusion on the north side of the lake. At Buchans, however, there is, as mentioned by Newhouse, an S-shaped bend in the structure for a distance of 8 miles along the strike. The ore bodies are near the center of the northward bulge of this curve. The structure at Buchans is gently folded. The Lucky Strike ore bodies are on the axis of an anticline plunging westward; the ore bodies at Old Buchans and Oriental are on the north limb of the same anticline, which at Oriental plunges eastward (Fig. 3). No important faulting is found near the mineralized zone, with the exception perhaps of fault crossing Buchans River between Oriental and Old Buchans, and striking N. 70° E. with dip 80° to the south. On the south shore of Red Indian Lake, several faults strike northeast-southwest, and there the rocks of Buchans series have been

subjected to considerably more folding, shearing and alteration than at Buchans.

Stratigraphic Section of Buchans Ore Zone

<table>
<tr><th colspan="2">Lucky Strike</th><th colspan="2">Oriental</th><th colspan="2">Buchans River Mine</th></tr>
<tr><th></th><th>Thickness, Ft.</th><th></th><th>Thickness, Ft.</th><th></th><th>Thickness, Ft.</th></tr>
<tr><td>Glacial drift.</td><td>5–40</td><td>Glacial drift</td><td>20–60</td><td>Glacial drift</td><td>0–55</td></tr>
<tr><td>Lamprophyre and diabase dikes.</td><td></td><td>Lamprophyre dikes</td><td></td><td>Lamprophyre dikes</td><td></td></tr>
<tr><td>Quartz porphyry and dacite[a].</td><td>0–350</td><td rowspan="6">(Quartz porphyry and pyroclastics removed by erosion)</td><td rowspan="6"></td><td rowspan="6">(Quartz porphyry and pyroclastics removed by erosion)</td><td rowspan="6"></td></tr>
<tr><td>Andesite, in places.</td><td>0–200</td></tr>
<tr><td>Quartz porphyry (and some dacite) partly sheared and altered.</td><td>0–60</td></tr>
<tr><td>Tuff and some quartz porphyry sheared and mineralized to high-grade sulphide-barite ore.</td><td>0–200</td></tr>
<tr><td>Quartz porphyry (and some dacite), partly sheared and altered.</td><td>0–100</td></tr>
<tr><td>Tuff and agglomerate with low-grade pyritic lead, zinc, copper mineralization in upper 150 ft.</td><td>600–800</td></tr>
<tr><td>Lava flow (basic, amygdaloidal) intercalated with several tuff beds and containing some low-grade sulphide and barite mineralization where thin sills of quartz porphyry have intruded the tuff beds.</td><td>100–950</td><td>(Lava flow, basic, amygdaloidal, 750 ft. east of Oriental)</td><td></td><td>Dacite in places</td><td>0–400</td></tr>
<tr><td rowspan="5">Tuff and fine-grained siliceous sediments with few beds of arkose. In Oriental haulage drift.)</td><td rowspan="5">Approx. 1000</td><td>Arkose</td><td>0–170</td><td>Conglomerate with matrix of high-grade sulphide-barite ore</td><td>0–17</td></tr>
<tr><td rowspan="3">Tuff and agglomerate with much intruded quartz porphyry. In the upper part the tuff and quartz porphyry have been sheared, altered and mineralized and contain bodies of high-grade sulphide-barite ore</td><td rowspan="3">0–400</td><td>Agglomerate or brecciated tuff</td><td>0–50</td></tr>
<tr><td>Quartz porphyry, in places</td><td>0–250</td></tr>
<tr><td rowspan="2">Tuff, and some arkose, the former containing high-grade sulphide-barite ore where quartz porphyry has intruded and where shearing followed the intrusion</td><td rowspan="2">200–250</td></tr>
<tr><td>Arkose and some fine-grained siliceous sediments</td><td>200–235</td></tr>
<tr><td>Lava flow (basic, amygdaloidal).</td><td></td><td>Lava flow (basic, amygdaloidal)</td><td>100–135</td><td>Lava flow (basic, amygdaloidal)</td><td></td></tr>
<tr><td></td><td></td><td>Sediments (siliceous, coarse and fine-grained)</td><td>840</td><td></td><td></td></tr>
</table>

[a] West of Lucky Strike, lava overlies the Lucky Strike quartz porphyry.

Section *C-B-A* on Fig. 3 is a longitudinal section approximately parallel to the following described axis. The part *C-B* is along the

folding axis at Lucky Strike, determined from geological cross sections and premineral jointing noted in the mine. The part *B-A* is along the strike of the folding in the northeast part of Oriental, as determined mainly by the direction of shearing observed. Actually the axis of the fold curves gradually east from Lucky Strike, where its direction is approximately S. 74° E. to N. 68° E. in the east part of Oriental. The actual location of the anticlinal axis at Oriental has not been definitely

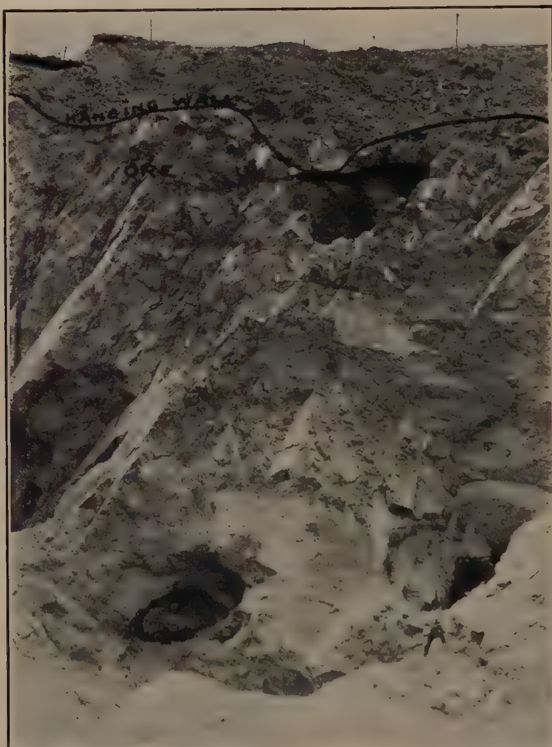


FIG. 7.—LUCKY STRIKE MAIN ORE BODY, GLORY HOLES.

Looking southwest towards the hanging wall. Note north-south jointing with dip of 60° to the east, normal to the plane of the fold which pitches 30° in westerly direction. There is also prominent jointing in east-westerly direction with dip averaging 75° to 80° to the north.

determined because of the overburden, but from sections it appears to be somewhere just south of the Oriental ore zone as shown on the plan.

Section *C-B-A* shows that the ore bodies at Lucky Strike are higher in the stratigraphic section than those of Oriental and Old Buchans, as the lava flow above the Oriental mineralized beds, which outcrops northeast of Oriental, underlies the Lucky Strike mineralization. Five sections at right angles and one at an oblique angle to section *C-B-A* (Fig. 3) are shown in Figs. 4 and 5. The location of these sections is indicated on the plan, Fig. 2.

Lucky Strike Ore Bodies

At Lucky Strike the ore bodies are found mainly in fractured tuff beds intruded by quartz porphyry (Figs. 2 to 4 and Fig. 6). The main ore body, which is at the center of the anticline, shows prominent north-south jointing with a dip of 60° to the east, normal to the fold; and also in east-westerly direction with a dip averaging 75° to 80° to the north (Fig. 7). Broad changes in mineralization occur between footwall and hanging wall along the axial plane of the fold.

Of the total of 5,300,000 tons of ore discovered at Lucky Strike, 85 per cent is contained in the high-grade main ore body, which outcrops under a shallow cover of glacial drift. The other 15 per cent is mainly contained in four lower grade bodies found on the north limb of the anticline, three of which, close together near the surface, are called the "North ore body," and the fourth, at a lower horizon, the "Two Level ore body." Over 90 per cent of the ore in the main ore body is lead-zinc ore; the remainder is classed as copper-zinc ore. There is no copper-zinc ore in the North ore body or the Two Level ore body. Assays of the ore are given in Table 1.

Table 1.—*Assays of Lucky Strike Ore*

	Oz. per Ton		Per Cent			
	Au	Ag	Cu	Pb	Zn	Fe
Main ore body:						
Lead-zinc ore	0.06	3.6	1.2	9.9	20.8	8.1
Copper-zinc ore	0.03	1.8	6.1	1.6	13.2	23.7
Average main ore body	0.05	3.5	1.6	9.2	20.1	9.4
North ore body	0.05	4.1	0.4	4.9	8.4	1.9
Two level ore body	0.05	4.4	0.6	5.8	9.7	2.8
Average total Lucky Strike	0.05	3.6	1.4	8.6	18.4	8.3

The lead-zinc ore mined to the end of 1935 from Lucky Strike has averaged: $BaSO_4$, 30.0 per cent; SiO_2, 4.0; CaO, 3.6; Al_2O_3, 1.4.

The lower grade bodies of lead-zinc ore listed in Table 1 contain from 50 to 60 per cent barite; high-grade sulphide ore may carry as little as 15 to 20 per cent. The main ore body consists of a dense, fine-grained intergrowth of sphalerite, pyrite, galena, chalcopyrite, barite and quartz, with minor amounts of calcite in fractures. The sulphides in the lower grade deposits of high barite content are somewhat less fine in grain. The gold and silver in the ore is mainly in tellurides (petzite and hessite?).

The average Lucky Strike ore contains from 0.002 to 0.004 per cent bismuth. The lead concentrates contain about 65 per cent Pb. and 11 per

cent Zn; they average approximately: Bi, 0.20 per cent; As and Sb, 0.05; Co and Ni, 0.10.

The main host rock replaced by the barite and sulphides is tuff. However, mineralization has also taken place in a small patch of conglomerate, in some sheared quartz-porphyry tongues and dikes crossing the main ore body, and in the andesitic agglomerate in the footwall of the ore bodies. The solutions carrying the barite, sulphides and tellurides have risen in the tuff along the contact with an overlying sheared and sericitized, thick quartz-porphyry sill. The shearing and hydrothermal alteration of the quartz porphyry extends in places for distances of over 20 ft. from the tuff contact. The bottom of the main ore body is at 300 ft. vertical depth below the surface; the North ore body is slightly shallower and the Two Level ore body extends from about 240 to 620 ft. vertical depth below the surface without change in character or grade of mineralization. Structurally, the Two Level ore body may be considered as a recurrence of the mineralization of the North ore body where the dip of the tuff beds changes from flat to 45° (Fig. 6). At greater depths (730 and 1775 ft.) tuff beds show baritic low-grade mineralization, of similar character, when in contact with thin sills of sheared quartz porphyry. Scattered low-grade, pyritic copper-lead-zinc mineralization, of lower barite content, extends for some 200 ft. into the tuff and agglomerate below the ore bodies, without any drop in sulphide content with depth. Roughly, there are some two million tons of this low-grade footwall material averaging: Au, 0.01 oz. per ton; Ag, 0.4 oz. per ton; Cu, 0.8 per cent; Pb, 1.9; Zn, 4.0; Fe, 12.3 per cent.

The south part of the main ore body along the hanging wall is partly bordered by 10 to 20 ft. of coarsely crystalline barite, colored red by hematite and almost free from lead and zinc sulphides. Analysis shows 83 per cent $BaSO_4$; the remainder is mainly hematite with minor amounts of calcite, chlorite and pyrite.

The ore in the main ore body varies much in character and grade. In general the footwall sections contain more sphalerite, chalcopyrite and pyrite and less barite, galena and gold-silver minerals than the parts closer to the hanging wall. The ore in the lower grade North and Two Level bodies consists uniformly of coarsely crystalline gray barite, replaced by various amounts of fine-grained sulphides. The gold and silver content is the same in the two lower grade ore bodies as in the main ore body. The ratios of zinc, copper and iron to lead compare as follows:

North and Two Level bodies.	1.70 Zn, 0.09 Cu and 0.41 Fe to 1.00 Pb
Main ore body.............	2.18 Zn, 0.17 Cu and 1.02 Fe to 1.00 Pb

The sequence of mineralization is indicated in Fig. 8.

Pyrite and barite were the earliest minerals introduced after silicification of the tuff. The heaviest pyrite mineralization is in the footwall sections of the main ore body, and in the tuff and agglomerate below

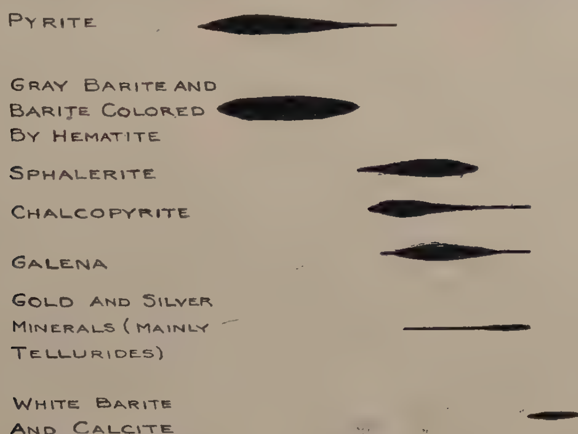


FIG. 8.—MINERALIZATION SEQUENCE AT BUCHANS.

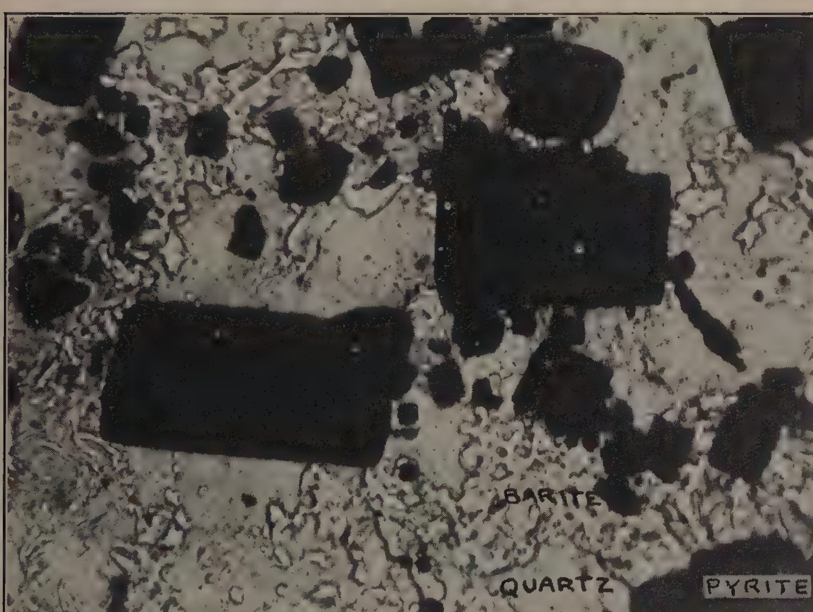


FIG. 9.—SILICIFIED TUFF FROM LUCKY STRIKE FOOTWALL, SHOWING REPLACEMENT OF QUARTZ BY PYRITE AND BARITE. ONE NICOL; × 360.

the ore bodies. Barite deposition is mostly confined to the ore bodies, but minor amounts of barite are found together with pyrite in the silicified tuff beds in the footwall (Fig. 9), and these also contain some small veins of gray barite slightly mineralized by sphalerite and galena.

The pyrite, which is by far the most coarse-grained sulphide in the ore, has been fractured and partly replaced by sphalerite and chalcopyrite. The pyrite fragments average about 50 microns in size. Fracture filling and mineral boundaries indicate that chalcopyrite replaced pyrite, sphalerite and gangue. Chalcopyrite is also found in fine, irregularly distributed specks in the sphalerite, frequently in size below 10 microns. Sphalerite replaced barite (Fig. 10), pyrite and quartz. In a few instances

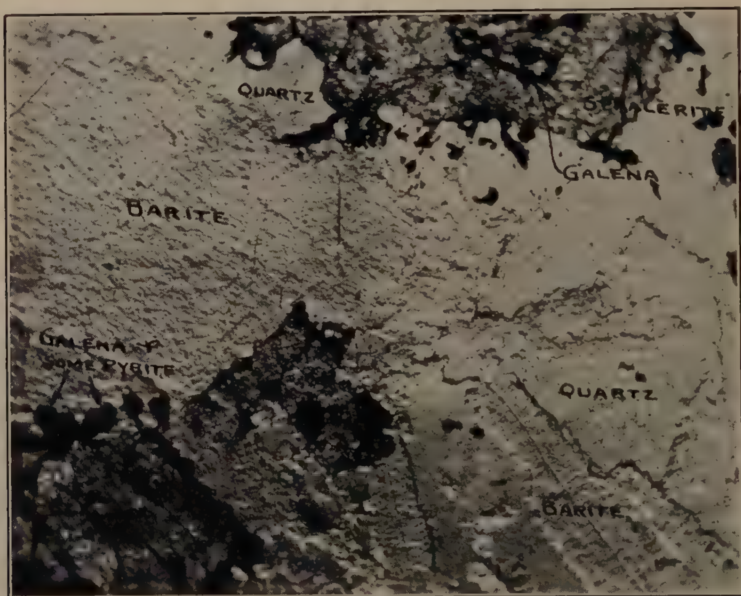


Fig. 10.—Mineralized tuff from Lucky Strike footwall showing replacement of quartz by barite, sphalerite and galena (black). One nicol; × 80.
Veinlets of barite cut the quartz. Sphalerite and galena replace barite in the lower half of the section, and galena fills fractures in the sphalerite. That the original rock was tuff is based mainly on field evidence.

replacement of sphalerite by pyrite has also been noted. Veinlets and mineral boundaries seem to indicate that galena partly crystallized simultaneously with sphalerite, chalcopyrite and pyrite, but that most of the galena replaced sphalerite and gangue. Galena frequently rims the sphalerite.

Chalcopyrite as well as sphalerite is found in greatest abundance in the pyritic footwall sections of the main ore body, whereas the galena content is higher in the parts containing gray barite as principal primary mineralization. The deposition of chalcopyrite and galena apparently continued until the end of sulphide mineralization, as some veinlets of slightly coarser chalcopyrite and galena cut the dense, fine-grained sulphides.

The gold and silver content of the chalcopyrite-pyrite-sphalerite ore in the footwall section is generally low. The highest precious-metal content is found in relatively low-grade lead-zinc ore near the sericitic hanging wall, where hydrothermal deposition probably continued later, to judge from the present water circulation, which takes place mainly along the hanging wall. The general distribution of the gold and silver minerals at least indicates that a considerable part of these minerals was deposited late in the sequence. The fine-grained gold and silver minerals have not as yet been clearly recognized in polished sections of the ore, but by observations and microchemical tests on selected grains from high-grade gravity concentrates it has been determined that they consist mostly of tellurides with minor amounts of native gold. Native silver has been found in a small specimen of calcite from one of the numerous veinlets of calcite and white barite which cut the main ore body.

Oxidation of the ore at the surface, and in fractures throughout the main ore body, has been rather extensive, particularly in the southeast part where the lead-zinc ore contains much pyrite and chalcopyrite. In parts the top of the ore body is altered to a gossan of high iron and lead content extending to a depth of from 5 to 6 ft. The sphalerite has leached away and some of its zinc content is found deposited as smithsonite in fractures below. The galena is partly oxidized throughout the main ore body and there is never less than 5 per cent of the total lead in the ore mined that is soluble in acetic acid. In a few months white films of water-soluble salts form on lead-zinc ore of high chalcopyrite-pyrite content. A sample of these white films, assumed to be sulphates although assayed for metals only, contained, if this assumption is correct: $ZnSO_4$, 64.1 per cent; $CuSO_4$, 32.5; $FeSO_4$, 2.9; $PbSO_4$, 0.5.

Oriental Ore Bodies

The Oriental ore bodies are confined to a series of weak, incompetent tuff beds intruded by sills of quartz porphyry, located on the north limb of the anticlinal fold where it pitches in an east-northeasterly direction. During the folding these incompetent rocks were strongly sheared and contorted by the differential movement between the relatively strong beds of arkose below and above the tuff. The ore bodies are irregular in shape and vary in size from a few hundred tons to a maximum of some 300,000 tons. The shearing and hydrothermal alteration of the quartz-porphyry sills have been more intense than at Lucky Strike, therefore much of the quartz porphyry adjacent to the tuff has been replaced by barite and sulphides to form low-grade lead-zinc ore. The soft quartz porphyry, sericitized and partly mineralized, and converted by dynamic action into a virtual schist, accounts for a higher average of silica in the ore mined at Oriental. The tuff, however, is the main host rock for the sulphides, as at Lucky Strike.

The ore bodies at Oriental are estimated to contain an aggregate of 2,220,000 tons of ore assaying as shown in Table 2.

Table 2.—*Assay of Oriental Ore*

	Oz. per Ton		Per Cent			
	Au	Ag	Cu	Pb	Zn	Fe
Lead-zinc ore	0.05	3.7	0.9	8.6	15.7	4.1
Copper-zinc ore	0.04	5.6	5.9	3.4	11.4	21.5
Average total Oriental ore	0.05	3.9	1.4	8.1	15.2	5.8

Ore Mined from Oct. 1, 1935, to Jan. 31, 1936

Oz. per Ton		Per Cent							
Au	Ag	Cu	Pb	Zn	Fe	$BaSO_4$	SiO_2	CaO	Al_2O_3
0.05	2.8	1.3	10.7	18.7	5.9	25.1	12.2	1.6	1.8

The ore is a fine-grained intergrowth of sulphides, barite and quartz. Considerable portions of the sulphides, and particularly of the galena and chalcopyrite, are in particles less than 50 microns in size and some grains are less than 10 microns.

The form of the ore bodies varies greatly. Lenticular or veinlike bodies are most frequent. The general strike of the ore is in east-northeasterly direction parallel to the main folding axis. The most frequent dip is about 45° to the north, but several ore bodies are almost horizontal and others dip about 45° to the south. Many of the irregularities in the shape, dip and strike of the ore have been caused by the irregular beds of andesitic agglomerate lying between the mineralized tuff and the lower arkose, or between tuff beds. The agglomerate, like the arkose, is fractured but has resisted shearing. It is partly pyritized and chloritized and shows scattered, low-grade lead-zinc-copper mineralization, similar to that of the agglomerate in the footwall of the Lucky Strike ore bodies. In the arkose, everywhere free from sulphides, the fractures are generally filled with calcite, but many are open enough for water circulation.

The character of the mineralization shows many changes parallel to the strike. The ore of high chalcopyrite-pyrite content, classed as copper-zinc ore, occurs mostly in bands up to 20 ft. thick within, or in the rims of the larger bodies of high-grade lead-zinc ore. Much of the pyrite is in separate bands in the walls of the ore bodies; it is not intimately intergrown with the sphalerite and galena to as great an extent

as in the footwall section of the Lucky Strike main ore body. In the largest ore body in the east end of the Oriental mine there is a band of coarsely crystalline barite, approximately 10 ft. thick, almost free from sulphides, and partly colored red by hematite.

The sequence of mineralization is in general the same as described for the Lucky Strike mine. In the ore formed by replacement of sheared and altered quartz porphyry, barite and some pyrite first replaced the quartz in the groundmass, leaving part of the quartz phenocrysts

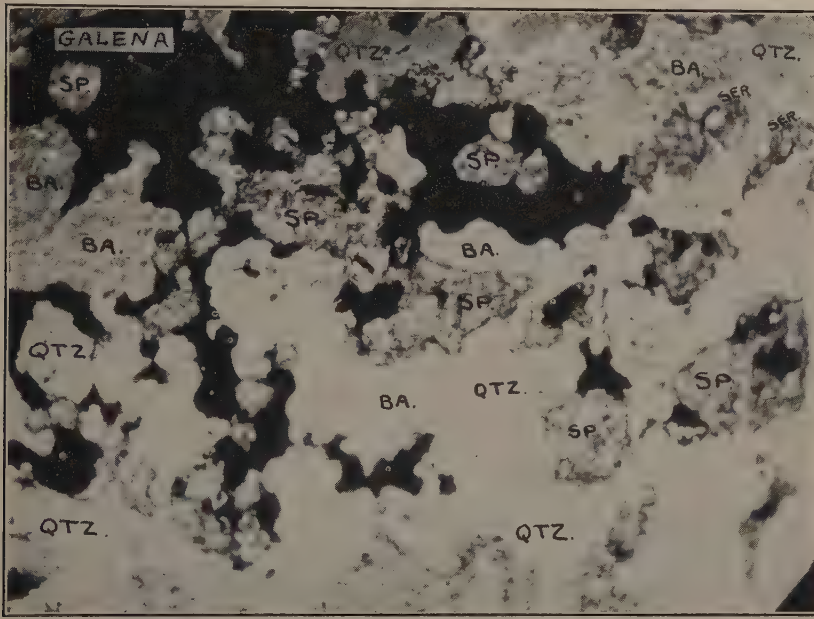


FIG. 11.—MINERALIZED QUARTZ PORPHYRY FROM ORIENTAL SHOWING REPLACEMENT OF PORPHYRY BY SERICITE (SER), BARITE (BA), SPHALERITE (SP) AND GALENA (BLACK). ONE NICOL; × 80.

Quartz is believed to be part of the original rock, determined to have been quartz porphyry, from field evidence. Note sphalerite in boundaries between quartz and barite, and sphalerite "corroding" barite in center of section; also galena cutting and "corroding" sphalerite.

and some sericite unaltered. Sphalerite replaced barite and quartz and sometimes galena; galena and pyrite replaced barite, sphalerite and quartz. (Fig. 11 shows a section of the groundmass of the sheared and mineralized quartz porphyry; Fig. 12 a section of the mineralized tuff from the Oriental mine.) Chalcopyrite is less frequent and the gold content is generally higher in the mineralized quartz porphyry than in the ore formed by replacement of tuff. The average silver content of the copper-zinc ore in the Oriental mine is 5.6 oz. per ton, as compared with an average of 1.8 oz. in similar ore from Lucky Strike mine. No determination has been made as yet of the minerals containing gold and

silver. It is probable that most of the gold occurs in tellurides, as at Lucky Strike.

The extent of the oxidation of the ore near the surface is still unknown. Some 200 ft. below the surface, however, 2 in. of bornite were found in a fracture in which surface water was flowing. In another part of the

Fig. 12.—Mineralized tuff from Oriental showing almost entire replacement of the quartz in the silicified tuff by barite, pyrite, sphalerite and galena. One nicol; × 80.

Note sphalerite in fractures and boundaries of barite in lower half of section; pyrite "corroding" barite and sphalerite in upper part to the right; also galena "corroding" barite. That the original rock was tuff is based mainly on field evidence

Oriental mine at the same horizon, calcite with native silver occurs in small fractures in fine-grained sulphide ore of high chalcopyrite-pyrite content. These fractures range in size from about 0.1 to 1.5 mm. The silver is generally in sheets less than 0.1 mm. thick, deposited in the boundaries between the sulphides and the calcite. The calcite occupies the center of the fractures.

DISCUSSION

(*E. S. Moore presiding*)

MEMBER.—How was the Two Level orebody shown in the cross section of the Lucky Strike mine discovered, since no drilling or shaft work appears on the section?

A. K. SNELGROVE,* Princeton, N. J.,—Recent studies by the Geological Survey of Newfoundland indicate that the Buchans formation was probably Ordovician and the igneous intrusion Late Silurian in age. This seems to tie up this region with European data.

P. W. GEORGE (written discussion).—The Two Level orebody shown in the cross section of the Lucky Strike mine was discovered by a crosscut driven northwest, 400 ft. below the surface, for exploration of the silicified and slightly mineralized tuff and agglomerate. The electrical prospecting done by Hans Lundberg in 1926 with such outstanding success in finding large orebodies had failed to show the existence of the Two Level orebody, presumably owing to its greater depth below the surface than the other ore deposits (240 to 620 ft.) and to its lower content of conductive, disseminated sulphides.

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Bedding-plane Faults and Their Economic Importance

By Charles H. Behre, Jr.,* Member A.I.M.E.

(New York Meeting, February, 1935)

Under the caption "fault," geologists intend to include all mass movements of solid rocks over adjacent rock masses. When these are studied long after their origin, however, circumstances make it possible to recognize as "fault" only formations showing clear evidence of actual movement of one side with respect to the other. Necessarily the chief evidence consists of the offset of a vein, bed, or other key structure. Planes of movement parallel to the bedding, or nearly so, are recognized with difficulty or are overlooked and hence many important faults are wholly unrecorded. Indeed, among standard American references on structural geology, one alone emphasizes the importance of bedding-plane movements and that one only briefly[1].

For some time the writer, cognizant of this difficulty, has paid special attention to such movements, both as observed in the field and as recorded in the literature. It is believed that enough data are now at hand to justify a short discussion. Its purposes are fivefold.

It is desired first to briefly classify and describe at least the more common kinds of bedding-plane faults, and second, to examine the causes and manner of their formation, at least in instances where the subject has not yet been well covered by other investigators. Third, certain by-products of bedding-plane movements are of such conspicuous importance to the economic geologist and mining engineer as to merit mention. Fourth, concrete illustrations should be placed before the reader in order to lend reality and to furnish a basis of comparison with his experience; such illustrations will necessarily be scattered through the text of this paper. They are only in part from the writer's own field observations.

The fifth and prime objective of this paper is to direct attention to the great importance of movements on the bedding planes, in the hope that the recognition of such movements and their significance may simplify mining problems.

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[1] References are at the end of the paper.

It is realized at the outset that many movements nearly or quite in the plane of the bedding grade insensibly into internal adjustments such as recrystallization or "flowage." These processes are essentially adjustments by atoms, molecules or crystal units, or at most small aggregates of crystals. This does not constitute true faulting and is not considered here.

Finally, on the basis both of field observation and of deduction, most cases of bedding-plane movement appear to be the result of compressional rather than tensional stresses. Exceptions are conceivable but only a few are known to the writer.

Classes of Bedding-plane Faults

Any plan for natural categories breaks down if due allowance is not made for the gradations between distinguishable facies. In offering the following classification, the presence of such gradations is not denied, but no purpose is served by avoiding systematization merely because of the occasional instances that resist classification.

The following six types of bedding-plane faults appear to be distinguishable, and significantly so; the terms used in designation of the types are not intended for introduction into the literature, but merely to serve as convenient abbreviations for use in this discussion: (1) saddle reef type, (2) steep reverse type, (3) transecting accessory type, (4) flat reverse type, (5) erosion type, (6) zigzag type. Each of these classes will be discussed in some detail.

Most faults are not unwarped planes, of course—indeed, they are anything but simple and planar. In view of their irregularity, it is not surprising that as the fault plane is followed downdip, especially in closely folded strata, occasional coincidence with the bedding planes should be observed; however, often this appears to be purely accidental and such instances are omitted from the classification suggested here.

Saddle Reef Type

The famous saddle reefs of Bendigo, at Victoria, Australia, may serve as type illustrations. Excellent descriptions of these structures were published by Rickard[2,3]. The true saddle reefs are auriferous quartz veins, which fill openings localized near anticlinal crests (Fig. 1). Some doubt may well be expressed as to whether the apical opening actually antedated the deposition of the vein matter, but there can be no question that the present veins represent an opening between the beds, rather than appreciable replacement.

Typically, the limbs of the folds associated with the saddle reef have fairly steep dips; frequently as much as 70°. Of two saddle reefs lying one above the other on the same fold, the lower generally has the wider

opening angle, the more gently dipping flanks, and less thickening on the crest. While fractures are found, conspicuous faults are rare and do not generally appear to affect the form of the reef itself.

Origin.—In order to develop openings at the crests of folds, it is obviously necessary for bedding-plane movements to take place. This type of movement is well established in geologic literature. There is some evidence in the same strata of local thickening (at the fold crests) and thinning (on the limbs); but individual laminae bordering the main bedding-plane parting can be traced respectively above and below the vein, which indicates that there was slippage along the surfaces of the country rock now immediately adjacent to the vein. This is clearly a case of bedding-plane movement. The cause for localization of slippage is not clear. It is probably the result in large part of differential elasticity as between different beds, but an analysis of this factor is not attempted here.

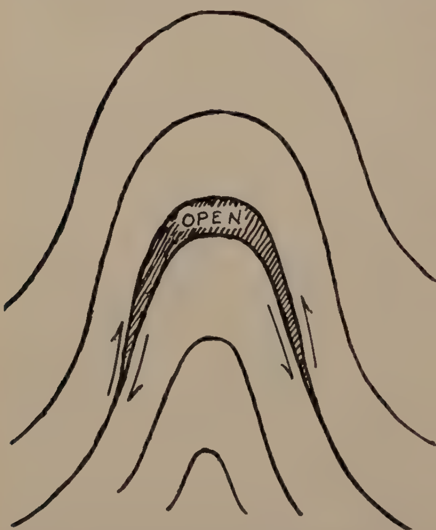

FIG. 1.—TYPICAL SADDLE REEF TYPE OF FOLDING, SHOWING MOVEMENTS ALONG BEDS.

Beds have flatter curves below reef or opening than above. Actual movement is confined to flanks of beds.

The difference in the elevations of the arch crests on the lower and higher beds separated by a given saddle furnishes a means for computing the relative updip movement of the higher beds. In cases available in the literature, this computation is not easily made because of lack of definite data as to the depth of the adjacent trough in a given bed. From one of the scale drawings presented by Rickard, the reef on the 2220-ft. crosscut of the 180 mine indicates an updip movement of 7.25 ft. in a thickness of 276 ft. of strata as measured at the crest of the fold; this computation is based on reasonable assumptions (Fig. 2).

Further light is shed on the progress of the opening to be filled by the saddle reef (or vein) by thickening of the strata at the fold crests, as previously noted. This demonstrates that internal adjustments preceded the actual tearing apart of individual strata and hence also preceded the bedding-plane movements discussed above. In short, a period of relative plasticity was followed by one of greater rigidity (probably the result of anamorphism), such as is necessary for bedding-plane movements. Metamorphism evidently did not destroy the bedding planes, however.

Finally, it is significant that small inverted saddle reefs, occurring in the synclinal troughs, have also been found at Bendigo. Adjustment takes place with a relative movement downward, therefore, as well as upward, but less conspicuously.

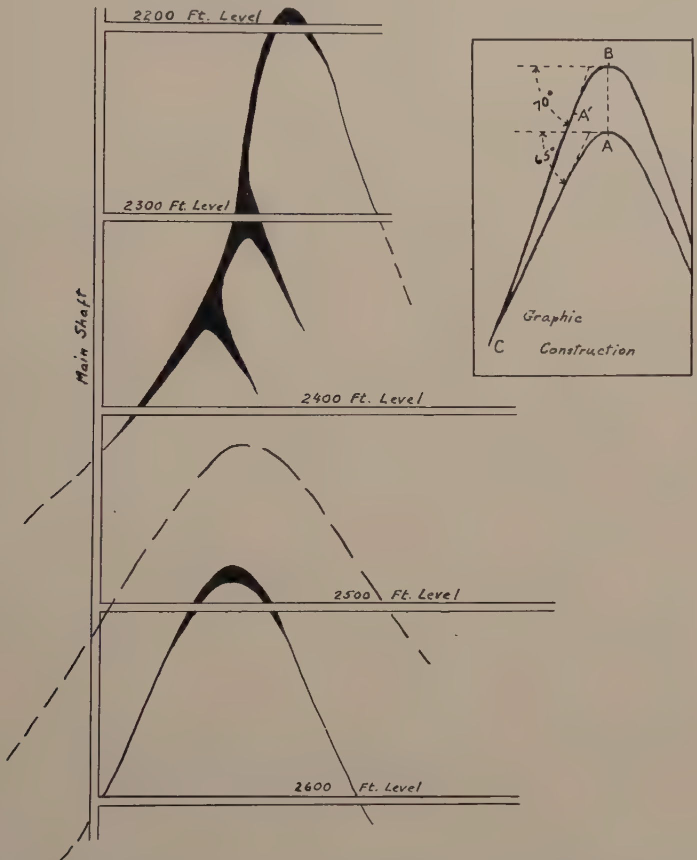

FIG. 2.—VERTICAL SECTION IN THE 180 MINE, BENDIGO (*after Rickard*).
To illustrate saddle reef type of bedding-plane faulting; note that upper folds are tighter than lower ones. Insert shows construction for graphic solution of amount of movement of upper bed CB over lower bed CA. A' is a point in space that, except for separation of upper and lower beds, would rest against A on lower bed; B is present position of crest point on upper bed; angles of dip of CB and CA are laid off respectively as observed from mine data, having been generalized with due regard to varying dips of beds as crest of fold is approached. AB is known from measurement; the problem is to measure $A'B$, the relative updip movement of bed BC.

Examples of Economic Importance.—As mentioned, it is not always possible to classify bedding-plane movements. When, however, in a region of fairly close folding they are neither manifest accessories of transgressive faults nor actually pass into such faults, they may perhaps properly be assigned to the saddle reef type. Examples of economic importance are numerous. The "flat fault" of the Ojuela mine in the

Province of Durango, Mexico, is probably an instance[4]. The ore deposits at Mascot, Tenn., seem to represent another[5] (Fig. 3); they occupy spaces in a breccia discussed below. Both Bateman[6] and Davidson[7] assign much of the localization of sill-like intrusions and also of the copper ores of Northern Rhodesia to bedding faults. Bedding-plane faults have even been found between basalt flows in the Columbia River region[8].

In the Rico district, bedding-plane deposits of uncertain origin have been described; they commonly bear ore and are spoken of as "blankets" or "contacts[9,10]." Some, at least, are certainly of tectonic origin, notably in the Rico-Aspen and Little Maggie mines, though others are regarded as solution effects.

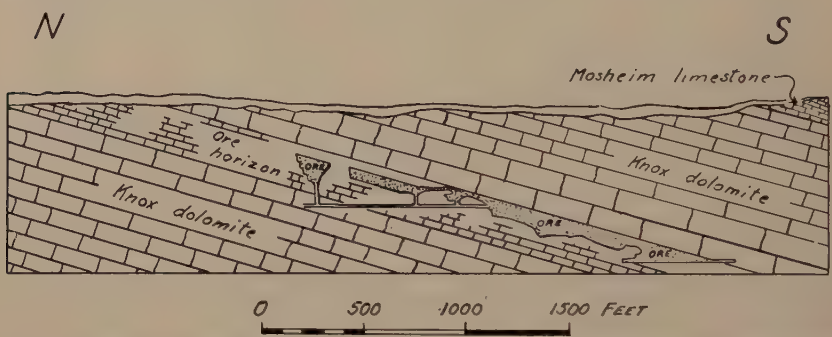

FIG. 3.—VERTICAL NORTH-SOUTH SECTION OF ORE ZONE AT MASCOT, TENNESSEE (*after Newman*).
Note that ore is confined to a single stratigraphic horizon.

Finally, saddle reefs, small but unmistakably showing the typical pattern, have been reported from the Philipsburg district in Montana[11].

By-products of Saddle Reef Type of Bedding-plane Movement.—Aside from the tendency for the beds to open along the separating planes, the saddle reef type of movement is likely to develop accessory fractures, not necessarily in the nature of faults, in the adjacent beds. Such fractures are figured by Rickard in the New Chum Consolidated, 222, and Lazarus mines[2] (Fig. 2).

Especially striking is the potential shattering of the more brittle rocks in one or both walls along the planes of movement. Such shattering is not necessarily confined to the anticlinal crests. A case in point is described by Hewett[12]; the limestone beds above and below the plane of movement are accordant, but they are separated by folded, thickened and shattered interbeds, also of limestone. These secondary features remind us strongly of the drag folds commonly developed under similar conditions in "incompetent" rocks between more rigid layers.

A further illustration of such shattering is to be found in the Mascot mine, already mentioned. The main ore bodies are at a fairly definite

horizon, dipping about 20° on the southwest limb of a broad fold of Cambro-Ordovician limestones. The ore occurs in a virtually continuous breccia zone[5,13], not clearly related to any single major fault (Fig 3). Currier suggests that such breccia horizons are due to shattering of the more brittle beds, but it is clear that he also recognizes the importance of bedding-plane slippage. These two processes are probably closely allied.

At Aspen, bedding-plane breccias in dolomitic limestones resemble those at Mascot. They have recently been restudied[14] and are attributed to movement between beds incidental to close folding; such movements are believed by Vanderwilt to be only measurable in inches, however.

Steep Reverse Type

Reverse faults may be derived from bedding-plane movements and at depth probably pass into such movements. If the picture drawn above for the saddle reef is correct, it may be expected that, as pressure from the sides continues, the fold will finally rupture at some point near the crest. The resulting fracture may constitute a nearly vertical reverse fault near the fold axis but, flattening downward, tends to pass parallel to and into the plane of separation between the beds (Fig. 4). Its vertical trace is concave upward. Downdip it is a true bedding-plane fault and its displacement in that direction may appear to be reduced to zero.

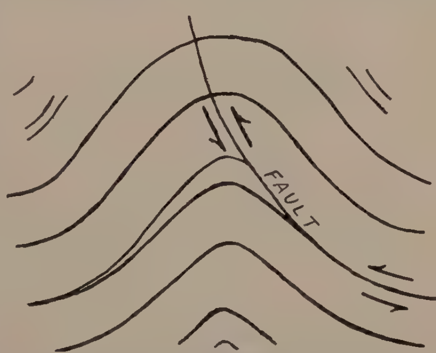

Fig. 4.—Ideal steep reverse type of bedding-plane movement, related to folding.

Mechanics.—In folds having essentially "parallel"* patterns, internal adjustment is negligible and noteworthy tangential compression is likely to result in faulting. In an anticline, if the overburden is great the tendency toward movement near the crest of the fold is largely taken up by internal adjustment; if the overburden is slight, however, as it is near the surface, the bed is more prone to behave as brittle material than at greater depth along the limbs, and hence fractures transverse to the bedding are to be anticipated.

In the troughs of synclines, on the other hand, faulting is less likely, for here the overburden (assuming erosion on the upwarped anticlinal area, concomitantly with folding) is still greater and flowage, as opposed to fracture, more probable. Moreover, lateral compression, in increasing the amplitude of the fold, forces it hard against lower beds, from which

* For use of this term, see Leith[15].

it encounters resistance that also favors internal adjustment and opposes fracture.

Thus, of the various parts of folds, anticlinal crests are least supported by contiguity with underlying beds, are least encumbered and urged into flowage by overlying beds, and therefore are favored points of fracture. Hence a bedding-plane movement tends to pass upward on a fold limb until it intersects the fold axis, whereupon the angle with the bedding increases and a recognizable reverse (or, rarely, normal) fault results. Conversely, a fault that at the surface is a reverse fault passes downward with reduced dip and finally coincides with the bedding plane.

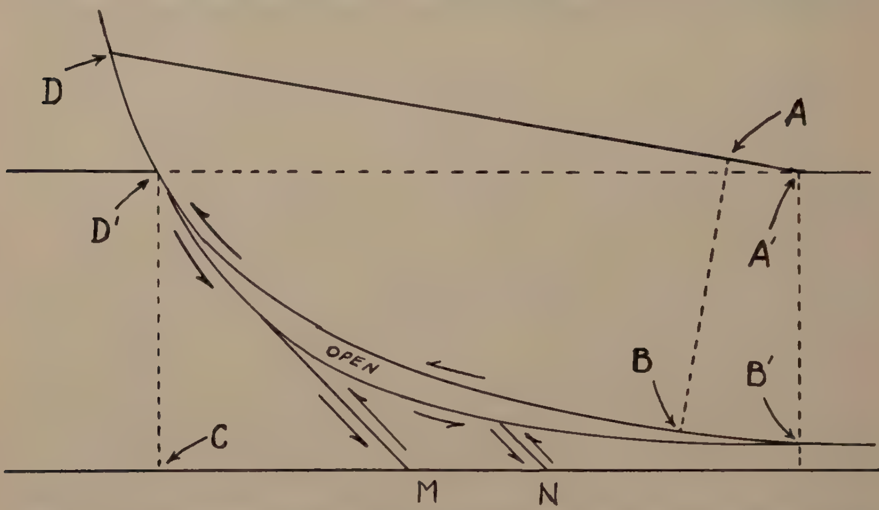

FIG. 5.—TO ILLUSTRATE "CHINKING IN" ALONG A CURVED MAJOR REVERSE FAULT PLANE.

Dotted lines outline an original block ($A'B'D'$) which, after the faulting, occupies the new position ABD. The roughly triangular block below the major fault (line $B'D$) has not changed its position. The consequent opening is likely to be filled up by reverse movement along the two minor and accessory faults at M and N.

Special Considerations and Accompanying Features.—Curved planes steepening upward are likely to result in considerable shattering, for reasons to be discussed next. Assuming vertical movement to be prominent, if the curve of the fault in a vertical trace is a perfect arc of a circle, movement along it does not destroy the matching of opposite sides. But if the curve is not strictly an arc of a circle, a space may develop. Continued pressure from the sides then tends to induce fracturing in one block or the other, and the small accessory faults that result have the effect of "chinking in" the opening. Their movement, if correctly deciphered, probably gives in most cases a clue to the movement on the major fault (Fig. 5), a fact suggested to the writer by others* but not, so far as he is aware, hitherto analyzed.

* T. S. Lovering and G. F. Loughlin, in conversation in 1929 and 1930.

While it is true that the steep reverse fault may be developed as a result of bedding-plane movements near synclinal axes, examples are rare. The writer figured one such case, the Bowden fault, as explored in the Ibex mine at Leadville[16]. Here, however, the opening to be anticipated at the trough of the fold was occupied by an intrusion.

Illustrations of Economic Importance.—In addition to the case just cited, several instances of this type of faulting are known from many mining districts. The pattern is not everywhere ideal, but usually conforms well enough to the description given above. In a north-south section through the Highland Boy, Yampa, and Utah Apex mines, a fold

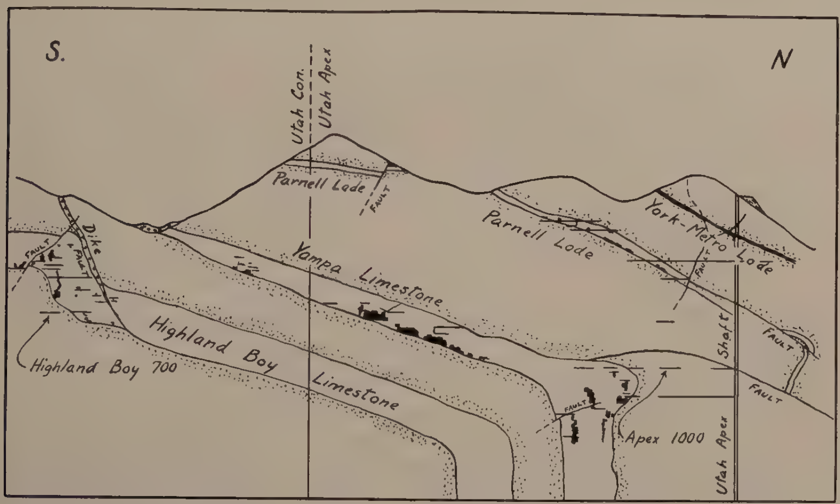

Fig. 6.—North-south section through Utah Apex shaft, Bingham, Utah (*after Hunt*).

To illustrate "fault folds"; one fault dips steeply north near south end of section and drops the limb of the fold; the other is slightly curved and almost horizontal, crossing Utah Apex shaft at about 1000-ft. level.

in the Highland Boy limestone is flanked by a slight fault, followed by a dike[17] (Fig. 6); these relations are unusual in that the fault is normal. Farther north a sudden westward steepening of the Yampa limestone yields a structure that may be regarded as an anticline overturned to the north. It is accompanied by a fault, which follows the upper surface of the limestone nearly to the crest of the fold; thence, it transects the beds. Other related cases are figured by Hunt under the term "fault fold" (ref. 17, Fig. 4 and p. 864). Such features, according to Hunt, "are obviously related in causal stress and in time to late cross faults from which they are structurally inseparable."

The great South Dyer thrust in the Leadville area, below which are localized a few small gold veins, brings pre-Cambrian upon Cambrian rocks (ref. 16, Fig. 1 and p. 46); for much of its extent, it is essentially

parallel to the bedding, and at its southeastern extremity it plays out into subsidiary thrusts that dip more steeply and transect the beds up the dip of the fault plane.

A modification of this type of movement is represented by several faults on the western edge of the great Tintic syncline between Eureka and Mammoth, Utah[18]. Here the beds dip very steeply east or stand vertical at the surface, but their dip flattens downward in approaching the synclinal trough; vertical faults, striking with the beds, cross the trough at depth and so offset the beds; the steepening of the bedding up the dip, however, soon brings bedding and faults into coincidence, so that the faults are largely unrecognizable at the outcrop, though they are readily found and their offsets measured underground.

Another special case that probably belongs here is that cited by the author from the Northampton County slate district, Pennsylvania. There, in relatively close folds, the outer, more tightly folded beds are cut off by fault planes parallel to the bedding on the inner, more open part of the fold[19]. As a result, much waste slate has to be quarried in the neighborhood of such faults, to the serious financial loss of the operators.

Transecting Accessory Type

This type of fault was first discussed, as far as this author is aware, by Emmons, Irving, and Loughlin[20]. It is associated with steeply dipping curved faults of larger displacement, and consists of movements on the bedding, the result of which is to permit adjustments along the major fault. An ideal diagram is presented in Fig. 7*A*, which shows both the main fault and the accessory movements along the bedding. Such faults are spoken of as transecting accessory, because they are movements accessory to larger faults that transect the bedding.

Origin.—Where a large fault traverses the beds and is curved in vertical section, the surfaces on either side of the fault plane must adapt themselves to such a curvature. If the rocks involved are so rigid that they do not flow readily, accessory faults may be developed and along these there is movement. If open, the partings along the bedding planes furnish surfaces of readiest movement. The result will be accessory bedding-plane faults like that shown in Figure 7*A*. Alternatives may be a relative movement of the beds in directions opposite to those already figured (see Figure 7*B*) or an actual shattering (Fig. 7*C*). Which of these possibilities occurs depends upon the curvature of the fault; thus, in Fig. 7*B*, a movement toward an arc of larger curvature (that is, one meeting the lower bed at a sharper angle than originally) will tend to cause movement of the upper bed toward the right and a leftward movement of the lower bed; and a bedding-plane fault may result under

these conditions also. The same general effect is that shown in Fig. 7*C*; here shattering with drag along the main fault plane has resulted.

It should be clear that bedding-plane movements of this type are favored by marked openings along the bedding planes and by the presence of relatively plastic rock, such as shale, between the more rigid layers.

Illustrations of Economic Importance.—At Leadville, bedding-plane movements of this type are common. Many of them even take place between such relatively rigid rocks as limestone and sills, especially where the limestone has been partly replaced by clayey minerals or the igneous rock of the sill has been altered. (See, for example, ref. 20, p. 65.)

At Weston Pass, not far from Leadville, is what appears to be a transacting accessory bedding-plane fault. It is near and dips toward the

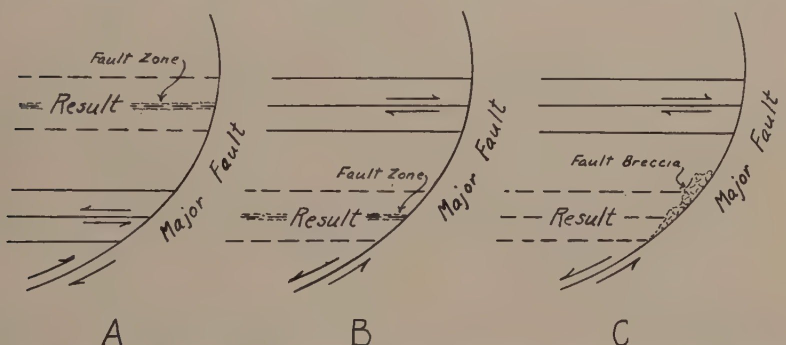

Fig. 7.—Ideal diagrams of bedding-plane faults of transecting accessory type.

Arrows show directions of movement on main fault and directions of movement on accessory bedding-plane faults. Solid lines represent position of beds before movement, dashed lines position of beds after movement. In *A*, the hanging wall moved up; in *B* and *C*, down.

Weston fault, one of the more important thrusts of the Mosquito Range, and is interpretable as accessory to the latter, though the geometrical relations between the two planes of movement are nowhere observable. The bedding-plane movement follows a definite stratigraphic horizon for a distance of about 7200 ft. and is characterized by the presence of a highly silicified breccia[21]. It is a striking fact that a virtually continuous ore horizon occurs about 80 ft. above this plane, suggesting the presence of a still higher zone of shattering similar to that just described[21].

In the Goodsprings, Nevada, district, the Argentena mine shows much ore in open breccia zones roughly parallel to the bedding of the limestones. Very close to the mine is the great Frederickson fault, a thrust dipping under the beds exposed in the workings; this important thrust fault shows movement largely horizontal and parallel to the strike of the fault plane (ref. 12, p. 148 and plate I). The breccia zones that bear the ore may well be accessory to the main movement on the Frederickson thrust. A similar bedding-plane fault with breccia zone is

described from the Anchor mine in the same district, but its origin and relationship to larger faults is not clear (ref. 12, 161).

One of the best examples of openings parallel to beds resulting from adjustments near faults is figured by Emmons from Philipsburg, Montana[11]. In this case, however, the faults involved appear to be normal. Here two small silver lodes rigidly follow two bedding-plane fractures between faults (see Fig. 8). Other similar cases are known from the same district (ref. 11, pp. 214, 215).

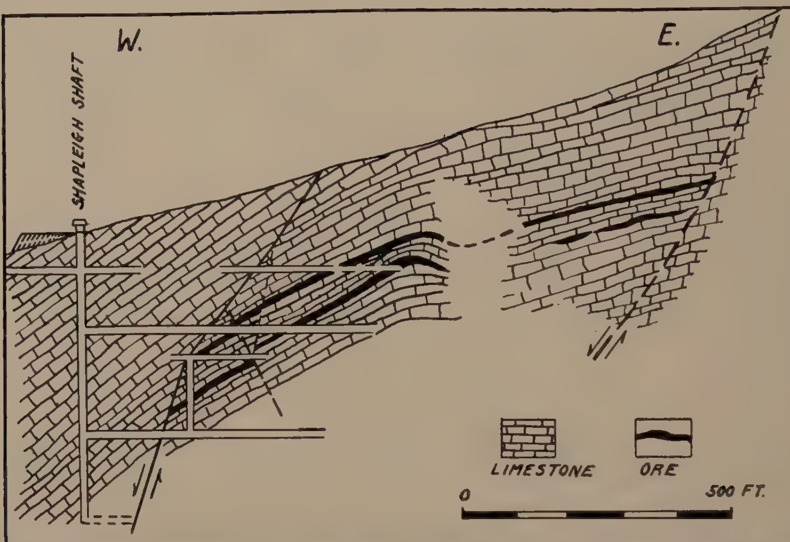

Fig. 8.—Vertical section of Hope mine, Philipsburg, Montana (*after Emmons and Calkins*).
Silver veins occupying bedding planes that appear to represent planes of movement accessory to faults shown.

Flat Reverse Type

In the flat reverse type of fault the fault plane dips very gently in the direction from which the thrust advances (underthrusting omitted, as a special case). Such faults are likely to show dips having angles of 15° from the horizontal, or less, and may, as in the folded thrusts observed in the Appalachians and in parts of the Rocky Mountains, even decline to zero, with occasional dip reversals. If the beds also dip gently in such instances, the fault and bedding planes may coincide. Many of the great thrusts belong to this class. The name used here to designate the type refers to the attitude of the fault plane and to the generally reverse nature of the movement.

Origin.—Faults of this type have long been the subject of study by tectonic geologists. An excellent discussion of their origin was presented by Chamberlin and Miller[22]. These authors point out that thrust faults flatten downward, on account of the effect of the overburden and

friction, which add a strong vertical component to the pressure; or, to put it differently, the planes flatten downward with increasing rotational stress. If the formation of the fault plane had not been preceded or accompanied by intense folding, the fault might at depth readily become parallel, or nearly so, to the bedding. This is also true where folding produces doming; in such a case, however, the trace of the fault on a vertical plane more nearly approximates that already described for the steep reverse type of bedding-plane fault.

Rich has recently illustrated such faulting in describing the Pine Mountain fault of southeastern Kentucky. He emphasizes its tendency to approximate a bedding-plane fault. Flattening is chiefly attributed by Rich to the presence of nearly horizontal layers of softer, more plastic shales. Oblique rises of the fault plane across the bedding are believed to result only where frictional resistance becomes very great; thereupon the fault plane rises more steeply to a higher level, on which it may meet another shaly series and follow this a way[23].

The two papers referred to discuss fully the problem of low-angle thrusting; they also show that such thrusts may tend to follow the bedding planes. Further consideration here is deemed unnecessary.

Examples.—In addition to the Pine Mountain fault, the planes of numerous large-scale and low-angle faults are, in parts at least, parallel to the beds. Examples include the Lewis thrust in Montana, and subsidiary smaller faults connected with it (ref. 24, Fig. 335 and Plate 53); the Heart Mountain thrust, at least in part as to the floor and very largely as to the hanging wall[25]; some of the flat faults of the Scottish Highlands (see, for example, ref. 26); and the Pulaski fault of the valley coal fields of Virginia[27]. The last named covers in part some of the most valuable coal beds of the state.

Excellent examples on a smaller scale, akin also to the steep reverse type of bedding-plane movement, are given by Reeves for areas bordering the Highwood Mountains. Here well-nigh horizontal beds show movement on the bedding planes; in areas of gentle folding such fault planes change to low-angle reverse faults that cross the fold axes. The movement mentioned is furthered by slippage on shaly beds[28]. It is of economic importance as bearing upon the origin of certain structures that may be petroliferous.

A special case, in which the thrust plane is steep but dips nearly parallel to the beds for an unknown distance, is described from an Arkansas cinnabar district[29,30]; the fault is on one limb of a compressed fold and thus bears some resemblance to the faults at Tintic cited above.

Accessory Details.—For the economic geologist one of the noteworthy features of major thrusts, whether or not they are parallel to the bedding planes, is their tendency to be accompanied by small accessory openings, sometimes in one, and frequently in both walls. In many ore deposits

the major faults themselves do not carry the ore; it is borne instead by these accessory fissures. In general, such minor fractures follow two different patterns; which one is followed depends upon whether or not they result from tension or shear. Accessory shear fractures make an acute angle with the major fault, an angle whose apex points in the direction of movement of the opposite side of the fault. Accessory tensional fractures (to which the term "gash vein" is also applied) yield wider crevices so oriented that the apex of the acute angle which they make with the main fault points in the direction of movement by the side in which such crevices are found (Fig. 9). The literature on theories of such features is voluminous. For particulars the reader is referred to any standard text on structural geology.

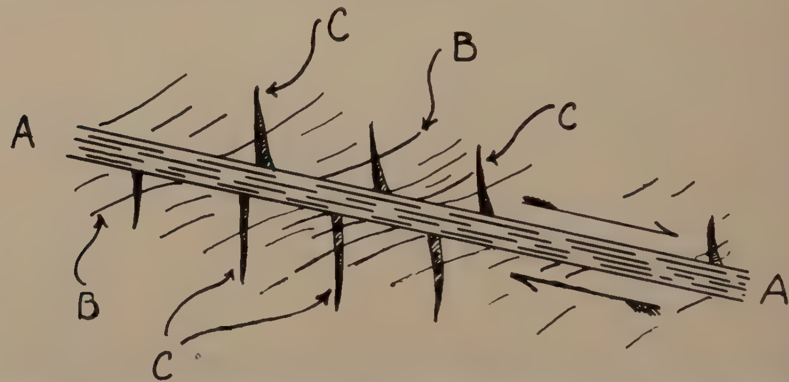

Fig. 9.—Idealized thrust fault (AA) showing relation between accessory shear fractures (B, B) and tension fractures or "gash veins" (C, C, C). Note that B, B are closed and C, C, C are open.

The formation of gouge is also a conspicuous feature, especially of most larger thrusts but of many minor ones as well. For this reason major thrust faults particularly frequently fail to show the extensive mineralization to be expected in view of their magnitude; they are tight barriers rather than channel ways.

The Erosion Type

The erosion type may be regarded as a modification of the flat reverse type just described. In it the fault plane coincides with a surface developed by erosion. Fortuitously the erosion surface and the bedding also coincide.

Origin and Illustrations.—Such cases seem to have been effected by a fairly constant sequence of events. There is first local bulging as the result of tangential compression. Gradually older, more rigid rocks are exposed on the bulge. Resistance on one flank of such a buttress is reduced by differential erosion, or by overfolding due to further tangential compression from the opposite side, or by both factors operating together.

In the meantime, a stripped plain or peneplain has been developed by erosion in the lee of the bulge. When pressure is renewed, the buttress breaks across the beds and advances over the erosion surface as a relatively flat thrust.

This sequence has been recognized in the southern Appalachians for the Rome and Cartersville faults (especially the latter) by Hayes[31] (Fig. 10); in the northern Rockies for the Lewis overthrust by Willis (ref. 24, pp. 339–340); and for the Heart Mountain thrust by Hewett[25]. The relation between this type of movement on an erosion surface (which perchance virtually coincides with the bedding) and the large-scale bedding-plane movement so commonly observed in the great overthrusts, is

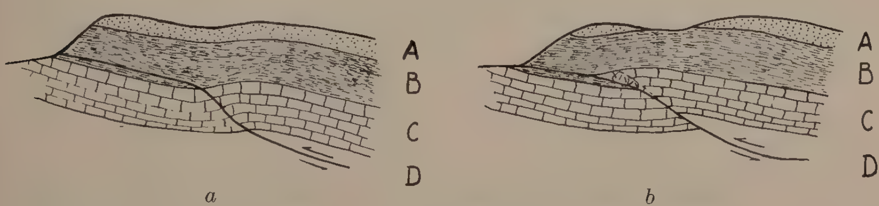

FIG. 10.—HOLLY CREEK SECTION OF CARTERSVILLE THRUST-EROSION FAULT (*after Hayes*).
a, before movement but after erosion; *b*, after movement. A, Cohutta conglomerate; B, Ocoee slate; C, Knox dolomite; D, Conasauga slate.

obvious. Accessory features in the two types are closely similar as well. Faults of this class are not generally important to the economic geologist except by virtue of accessory features.

Zigzag Type

The vertical trace of a compressional fault sometimes approximates what may be called a zigzag pattern. In such instances the fault plane follows the bedding for some distance, then turns at an abrupt angle and cuts obliquely across the beds, only to follow them again at a higher or lower level. In the larger thrust faults this pattern can seldom be traced, partly because continuous sections along the fault planes are lacking, partly because the irregularities are curvilinear rather than sharply angular in the manner just described. In smaller faults, however, where the vertical displacement is only a few feet, the zigzag pattern is fairly common, as examples will show.

Illustrations.—So far as the writer is aware, Barrell first called attention to such fault patterns, reproduced on a small scale by shear planes and bedding faults, which he concluded were jointly responsible for local mineralization at Marysville, Mont.[32] Another illustration, on a larger scale is cited by Rickard in the Enterprise mine, where a fault transects at a high angle beds that are virtually horizontal up to the level of a

certain limestone layer, in which it follows an approximately horizontal course parallel with the bedding for a few feet, and then again cuts across the beds (ref. 33, pp. 942–946).

A case from Philipsburg, Montana, has been well figured by Emmons and Calkins, under the term "roll" (ref. 11, p. 255, Fig. 54).

At Leadville, Colo., there are also numerous illustrations of the zigzag type of bedding-plane fault. One of the best is in the Oro La Plata mine, where, according to Argall, a bedding-plane fault fissure in the "Blue" limestone bears lean ore for a distance of about 500 ft. up the dip; then it turns to rise across the beds at an angle of about 30° and becomes wider, carrying richer ore in this part of its course. In this upper,

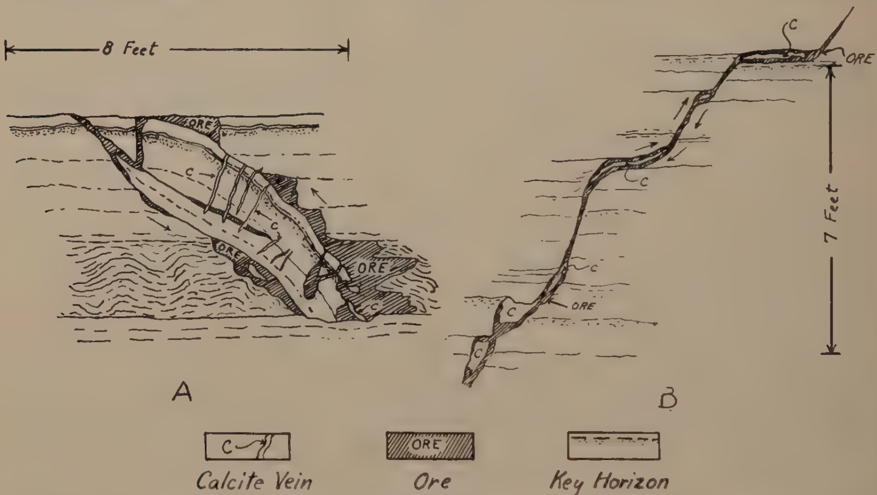

FIG. 11.—DIAGRAM OF TWO VARIETIES OF ZIGZAG BEDDING-PLANE FAULTS AS SEEN IN CRAWFORD MINE, HAZEL GREEN, WISCONSIN.

A, thrust fault in southwest workings (*after Scott*). Fault had course in lower, crumpled, shaly layer, then split upward so that its two branches encircled block, and finally continued to left just beneath topmost bed. Key horizon clearly shows compression, together with rotation of loose block.

B, thrust fault in northwest heading. Key horizons show that left (or hanging) wall has risen, giving bedding-plane partings now filled with ore; hence veins are wider where fault plane is horizontal.

inclined part the displacement is clearly discerned and the dip-slip amounts to about 10 ft. This is a reverse fault[20].

In the lead-zinc district of southwestern Wisconsin, the ore bodies as now worked are chiefly "flats" yielding sphalerite. Closely examined, they show little if any replacement; instead they constitute tabular fissure veins, frequently several hundred square feet in plan though only a few inches or at most a foot or two thick. Careful study shows that at least some of the flats are related to bedding-plane faults of the zigzag type. The plane of movement generally follows bedding partings in the more thin-bedded or shaly lower Galena dolomite, rises obliquely across

a higher, more massive layer, and then, instead of passing into the roof along its projection updip, once more turns to the horizontal position and follows the bedding. Scott has described and figured several such cases[34] (Fig. 11).

It is to be expected that in such fractures a movement prevailingly parallel to one of the limbs of the zigzag may serve to widen the fault fissure, thereby separating its sides, while conceivably closing the fissure on the other limb—a principle familiar to mining geologists. (See for example, Emmons[35].) In this district horizontal openings preponderate; they exhibit considerable horizontal extent and generally appear immediately above a prominent shaly zone having the high plasticity that would favor bedding-plane movement. This general association supports the conclusion that horizontal compression, by producing differential movement along zigzag faults, is of far greater significance in the ore deposition of the district than hitherto realized. Indeed, a careful matching up of the vertical displacement with the vertical thickness of the "flats" shows close correspondence between the two.

Origin.—Such features are obviously the result of varying brittleness and plasticity of the materials in a normal sedimentary sequence. Thrust faults thus break across the beds not in an unwarped plane but in a fracture of which the inclination to the horizontal increases with increasing rigidity of the material traversed. If the beds are steeply inclined, faults resembling those at Tintic are more likely to be formed. But where the beds are flat and there is thrusting without a very heavy overburden, fault planes that otherwise would have dips of 45° or so are resolved into two components by the variations in the brittleness of the beds traversed. Since the direction of easiest relief is upward, the hanging wall moves up, rather than forward, and the parts of the fault that appear as more horizontal fissures are more prominent than those more steeply inclined.

Uncertain Classification

In many instances, if not in most, there is uncertainty as to the class to which a particular bedding-plane movement should be referred. Below are listed several examples, which, nevertheless, have been highly important in exploring the respective deposits. In these instances the associated structures, which usually are the main criteria in the classification scheme outlined here, are either not definitely known or not sufficiently described in the literature to permit positive classification.

In the Gogebic Iron Range of Wisconsin and Michigan several bedding faults have been described by Hotchkiss. These faults occur along one or more shear planes, usually in the Yale member, a series of ferruginous cherts and slates. The aggregate movement on this plane or planes is known and the major component is parallel to the strike of the fault. Thus, along the main fault in one locality the displacement amounted to

800 ft. along the strike of the fault plane and 200 ft. up its dip. From the succession of events as worked out by Hotchkiss, it appears that this fault is essentially a flat thrust, antedating folding[36] and thus not assignable to mere bedding plane adjustments of the Bendigo type.

In the Tri-State lead and zinc district, Fowler and Lyden have ascribed to movement on the bedding planes the openings of the "sheet ground" deposits. These are said to be the result of slight but recognizable tangential adjustments that characterize the district as a whole[37].

On a far smaller scale, bedding-plane faulting is shown at Rico, Colo., by the lateral shifting along the bedding of small cross veins (ref. 33, pp. 922–923, 946).

Conclusions

It appears possible to classify the types of bedding-plane faults on the basis of their structural associations and genesis, with due regard to gradations between types. Moreover, there are numerous illustrations available in the literature, especially in regions where evidences of lateral compression are definite or inferred. And finally, many of the cases are of sufficient economic significance to justify the conclusion that bedding-plane faulting should be ranked among the important structural features associated with ore deposition.

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Economic Application of the Insoluble-residue Method*

By H. S. McQueen,† Member A.I.M.E.

(New York Meeting, February, 1936)

The insoluble-residue method for the examination and correlation of limestones and dolomites, or other sedimentary rocks containing calcium and magnesium carbonates, originated and was developed in the laboratory of the Missouri Geological Survey. The numerous and ever-increasing problems in the field of applied geology, and the need for a tool or aid in the solution of them, led to experimental work with samples collected from deep wells, in the early part of 1924. The success attained, almost at the outset, with this method was so great that the method has now been developed from the state of the experimental to the realm of large-scale, every-day routine.

In a previous paper the writer[1] described the procedure in detail and the characteristics of the residues obtained from certain Paleozoic formations in Missouri. The purpose of the present paper is to discuss the economic application of the insoluble-residue method.

Developmental History

Within the Ozark region of southern Missouri the geologic column consists of Lower Ordovician (Canadian of Ulrich) and Upper Cambrian formations, which are composed mainly of dolomite with subordinate amounts of sandstone, shale and limestone. Chert is found to some extent in all except one of the dolomite formations. The stratigraphy of the region is complex because the dolomites have lithologic similarities, fossils are few and often poorly preserved, and identifiable horizons within the comparatively thick formations are not common. Although considerable stratigraphic work of an areal nature had been done for a period of many years by the Missouri Geological Survey, the detailed geologic succession had not, until a few years ago, been systematically worked out. The problems of subsurface stratigraphy were even more complex.

This region constitutes the main ground-water province of the State of Missouri and within it wells of varying depths are drilled in order to obtain supplies of ground water. In the work of the State Geological Survey in connection with the drilling of water wells for public supplies

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[1] References are at the end of the paper.

it was difficult to properly identify formations, particularly those composed of dolomite, as they appeared in well samples. It was also difficult to render opinions regarding the formations being drilled, whether the well should be continued in depth, and whether certain aquifers had been encountered or were absent.

A considerable portion of the earlier regional stratigraphic work was done by E. O. Ulrich, that in more recent years by Ulrich, H. A. Buehler, C. L. Dake, Josiah Bridge and the writer. Ulrich originally proposed the idea that certain formations, upon weathering, produce distinctive types of chert, and the principle was utilized successfully by him in field work. In the study of samples from deep wells within the region, it appeared that the same principle might be used on a microscopic scale, therefore samples from wells were digested with dilute hydrochloric acid and the resultant insoluble fractions, or residues, were studied. Some of the residues were very striking in appearance and contained elements that had not been observed before in the field. As the investigation continued, specimens collected from outcrops were also digested with acid and it was apparent that the correlation between well samples and outcrop samples from a formation could be made with confidence.

Mode of Procedure

The procedure in the preparation of an insoluble residue was described in some detail in the author's earlier paper, therefore, as no general changes or improvements in technique have occurred, the procedure will only be outlined briefly here.

Samples from wells are treated without respect to size of the individual particles. A sample, approximately 25 to 30 grams, is placed in a 250-c.c. Pyrex beaker. The sample is covered with approximately 50 c.c. of commercial hydrochloric acid, previously diluted with an equal amount of water. When the insoluble residue is needed for immediate study, the sample is heated on a hot plate in order to accelerate digestion of the limestone or dolomite; if speed is not necessary, the sample is digested without heating. After effervescence has ceased, the beaker is filled with water and the contents are allowed to settle for a very short time. Decantation by hand follows, and the fines, which consist of clay or silt, are poured off. Then 50 c.c. of diluted acid is added and the process of digestion and decantation is repeated. If all the carbonates are not removed by the two treatments, the sample is treated a third time. Except for a few dolomites, no further treatment is necessary. The sample is again washed and all the clay or silt particles are carefully removed. The sample is then placed on a sand bath or hot plate and dried, after which it is placed in a properly labeled, small glass vial.

In the preparation of insoluble residues from outcrop samples, the same procedure is followed although usually the samples are crushed

to a size that will pass a 10-mesh screen. When a sample is known or suspected to be fossiliferous, much larger fragments are utilized in order to obtain any fossil fragments that might assist in stratigraphic determinations. In the preparation of such residues, the fine silt and clay fractions are saved for future study.

The weighing of each sample taken for study, and, following digestion, the weighing of the individual fractions thereof, has not been practiced in the laboratory of the Missouri Geological Survey. In any detailed studies of field specimens of carbonate rocks, however, weighing of the original and the resultant fractions is no doubt worth while. In the preparation of insoluble residues from well samples for the purpose of quickly determining the formation being drilled in a well, the volume method, as practiced, has been found satisfactory.

A number of papers have been published on the subject of insoluble residues, and in general the method of preparation is the same as that outlined above, except the method described by St. Clair[2], who found the use of acetic acid more satisfactory in the study of Upper Silurian limestones found in New York.

Local and Regional Application of the Method

Within a period of some 12 years, 82,000 samples of insoluble residues have been prepared and studied in the laboratory of the Missouri Geological Survey. The outstanding contribution made to date as a result of this study is that each formation or stratigraphic unit yields, upon treatment, insoluble residues different in one or more ways from those obtained from other formations. It is worthy of note that certain types of insoluble residues, although comparatively limited vertically, have a wide lateral or regional distribution, and it is not strange, as it may seem on first thought, to find sections separated by many miles and in entirely different geologic provinces, similar in so far as the general characteristics of the residues are concerned. This statement is made with the utmost confidence and is based upon results obtained by the writer in a study of insoluble residues prepared from well samples collected in Arkansas, Illinois, Iowa, Kansas, Kentucky, Michigan, Missouri, Montana, Nebraska, Nevada, Oklahoma, Tennessee, Texas and Wisconsin.

The residues prepared from the Upper Cambrian formations (as restricted by Ulrich) in the upper Mississippi Valley are strikingly similar to those from the central portion of the Valley, and both possess general characteristics that again are exhibited in the Upper Cambrian residues from the Arbuckle Mountain region of Oklahoma and the central mineral region of Texas. The formations of the proposed Ozarkian system likewise possess features that designate them throughout the central and upper Mississippi Valley, and extend without great change or modification of type into the northern Mid-Continent region. Lower Ordovician

(Canadian) residues from eastern Tennessee and southern Kentucky possess the characteristics of the same residues in Missouri, the upper Mississippi Valley states, and Arkansas, Kansas, Oklahoma and Texas. The possession of diagnostic characteristics, and their regional distribution, has also been noted in comprehensive studies of Ordovician, Silurian, Devonian and Mississippian limestones and dolomites from the central United States.

As a result of these regional studies, the writer may state, without hesitation, that systemic classifications on the basis of general characteristic of the residues are definitely possible. The following example of the regional occurrence of almost identical residual characteristics, and the identification of them, is presented in substantiation of the foregoing statements. Field specimens were collected with care, and from carefully measured sections in the central mineral region of Texas, by the late C. L. Dake. These were submitted to the writer and insoluble residues were prepared and studied. The same sections were studied later in the field by Dake and Josiah Bridge, of the United States Geological Survey. The former, by virtue of his regional experience in stratigraphic geology, had drawn certain formational contacts, and the latter had confirmed them from the study of the fossils contained in the rocks.

Later still, Bridge and the writer held a conference at which the results of the individual studies were compared. It was found that the determinations made by means of the residues, both from the standpoint of systemic classification and the zoning of the sequence of beds involved, corresponded closely with the results obtained from field and paleontologic studies.

In addition to the foregoing, details have now been obtained for many formations in Missouri whereby it is possible to separate comparatively thick formations of dolomite or limestone into zones. In certain instances, some zones are locally absent and others thicken and thin. Thus, the breaking down of formations into zones, and the extension and correlation of them, has assisted materially in the correct interpretation of local and regional stratigraphic problems.

Economic Application of the Method

The insoluble-residue method has been employed in Missouri chiefly in connection with the drilling of water wells. In many localities in the southern part of the state, the geologic section consists in the main of formations composed of dolomite and chert from which varying supplies of water are obtained through crevices and openings. In earlier studies of the ground-water resources of the region information was collected to the effect that these water horizons occurred in the dolomite, but no attempt was made to correlate them from one locality to another, nor

was it known whether they were local or attained regional distribution. As a result of studies by the method under consideration, it has now been found that certain persistent and productive water-bearing zones occur at the unconformable contacts of formations consisting of dolomite. Intraformational contacts have also been definitely established in the Ozarkian dolomites, and they, too, are sources of water.

Three formations containing sandstone, in whole or in part, are present in the Ozark region, but in two of them at least the sandstones are locally absent or become dolomitic and as a result yield no water in important amounts. In areas where the sandstones are locally absent, the fact can be determined by the insoluble-residue method without question, and decisions made as to the procedure necessary to encounter the next possible point of production.

The knowledge, therefore, is now applied extensively by the Missouri Geological Survey in connection with the drilling of water wells, and as a result wells have been completed successfully in recent years, whereas they might possibly have been abandoned at shallow depths and without obtaining adequate quantities of water, or else drilled blindly and without specific guides to greater depths.

All wells drilled in Missouri for public supplies must be cased at the point determined by the State Geological Survey, and a study of the residues is particularly helpful in this connection. The presence of large amounts of chert in certain formations is suggestive of conditions of open ground, and indicates that a particular well should be cased below that point in order to protect it adequately from contamination by surface waters. The residues are also guides in determining the depth to mud-filled "open ground" or crevices which are common to and occur at specific horizons in some dolomite formations in the north central Ozark region of Missouri.

The use of the insoluble-residue method in the field of ground-water hydrology is not confined to Missouri, for in recent years it has been used by the Illinois Geological Survey Division (letter from M. M. Leighton); by the Iowa Geological Survey (letter from A. C. Tester); by the Wisconsin Geological Survey (letter from F. T. Thwaites); and by the Nebraska Geological Survey (letter from G. E. Condra).

The method is adapted to the drilling of wells for oil and gas, and has been successfully used in that connection. It is of utmost economic importance to identify properly the formation in which oil or gas occurs in areas where production is obtained from limestone or dolomites, or from other rocks associated with them. The writer's regional experience has indicated that the method can be applied with success in certain areas at least in the Mid-Continent region, where unconformities of considerable magnitude are present and add to the complexities of the geologic section.

The following specific uses have been made of the insoluble-residue method in stratigraphic studies in and near areas producing oil and gas. Martin[3,4] has reported upon studies of the insoluble residues from Mississippian limestones in Indiana and Kentucky. According to Walter F. Pond (personal communication), the same author has also studied the Chester formations of Mississippian age in Tennessee, with good results. The report has not yet been published.

Ockerman[5] states that the method has been useful in connection with a study of the Hunton and Viola limestones in the Forest City and Salina basins in Kansas. Both limestones produce oil elsewhere in that state. T. C. Hiestand (personal communication) has also utilized the method in oil and gas investigations within the same state. Eddy[6] has reported upon insoluble-residue studies of the Traverse and Dundee, oil-producing formations in Michigan, but R. A. Smith, State Geologist of Michigan, states (personal communication) that the results achieved with the method in Michigan show "that insoluble residues will help materially in correlations over small areas and perhaps occasionally in larger, but as yet cannot be used in state-wide correlation of formations." He also says that some 10,000 residue samples have been prepared from these formations for future studies, which "may change our present tentative opinions."

Ordovician rocks have been studied in some states with reference to oil and gas possibilities. Walter F. Pond, State Geologist of Tennessee, mentions (personal communication) that some studies of them have been undertaken in that state. The results of studies of the Ordovician rocks of Kentucky have been discussed by Meachem[7].

Stratigraphic work in central Kansas by means of the insoluble-residue method has been described by Koester[8].

A direct application of the method has recently been described by Burpee and Wilgus[9] for the Hobbs and Eunice oil pools in Lea County, New Mexico, their studies being utilized in local exploitation problems, such as uneven encroachment of water, in acidizing productive zones, and in determining the stratigraphy of the producing zones.

In the field of mining geology the insoluble-residue method has been successfully applied to problems existing in areas where mineral deposits occur in limestones and dolomites. In Missouri, it has been used in a detailed study of the Bonneterre (Upper Cambrian) formation, which carries the disseminated lead deposits of St. Francois and Madison counties. The study has indicated that the ore occurs within a definite zone in the formation. Some distance from the district this zone is absent, and also, not far from the producing district, other zones, not present in the district, occur within this formation. Suggestions relative to the depositional conditions under which this formation were laid down have also been obtained. Insoluble residues have been used in the

Tri-State lead district, where the ores occur in cherty limestones of Mississippian age, which lend themselves exceedingly well to this method. The stratigraphic succession and, in some instances, intraformational zones, have now been worked out and correlated throughout this district.

The insoluble-residue method has been applied recently by Singewald and Reed[10] in connection with the study of the geology and ore deposits of the Alma district, Colorado. In this locality this method gave them a means for identifying the Leadville limestone, in which "the chances of finding rich silver-lead deposits beneath the Weber (?) formation are much greater . . . than in any other formation."

Perhaps the outstanding use, however, in connection with mining is by the American Zinc, Lead and Smelting Co. at Mascot, Tenn., where zinc ores occur in the Cotter and Jefferson City members of the Knox dolomite. Several years ago the writer studied, in the laboratory of the Missouri Geological Survey, insoluble residues prepared from samples of drill cuttings from this locality and found certain striking characteristics. The method was adopted by the company mentioned, and the results obtained by them in the investigation have been described in a recent personal communication from M. H. Newman, geologist of the company, a portion of which follows:

> Studies of the insoluble residues from churn-drill holes in these two formations have indicated the presence in both (the Cotter and Jefferson City members) of individual beds that yield sufficiently characteristic residues as to permit of satisfactory correlations by comparison, and by the noting of stratigraphical intervals between beds. Apparently these conditions are rather constant, at least over considerable areas adjacent to the mines. Correlation in such detail has resulted in greater accuracy in the determination of the proper depths of drill holes, and has added assurance in that respect. It also has added much to our impressions of the characteristics of the beds that make up the formations drilled.
>
> The use of insoluble residues here is confined almost wholly to the study of churn-drill cuttings—it has been found to be more effective in deep than in shallow holes, as much dependence in correlation is placed upon intervals. Its use has been made a part of the routine in churn-drill prospecting and it is looked upon as an important adjunct to that work—as time passes more dependence is being placed upon its use.

In an ore-dressing investigation of the disseminated lead deposits of southeast Missouri, conducted by the United States Bureau of Mines, insoluble residues were made from specimens of ore-producing dolomite. A study of them shed light upon the relation between the mineral particles and the other constituents of the rock.

The insoluble-residue method has also been applied in the field of the nonmetallics, and was employed in a study of limestone deposits in St. Louis City and County, Missouri, by the late Charles D. Gleason[11], who found the method to be the solution to the problem of determining the contacts between the Ste. Genevieve, St. Louis and Spergen formations of Mississippian age. Further, the method showed that the charac-

ter and quantity of the residues are an index to the value of the limestones from these formations for use as concrete aggregate.

The method has been used for a similar purpose by the Illinois Geological Survey Division, the results being described (personal communication) by Dr. M. M. Leighton, Chief of the Division, in part as follows:

> We have used the insoluble-residue method with excellent results in a special study of the lithologic characteristics and correlations of the Silurian formations. By the determination of the quantities of residue as well as their characters, we have been able to assign all outcrops and quarries of Silurian rock in the northeastern part of the State to their proper position in the geologic section. As a result, the structural geology of the region has been revealed and it has become possible to determine the geographic locations of desirable stone not yet quarried. The proportions of insoluble residues have been found helpful in determining the relative economic importance of the rock at various horizons also. The importance of these determinations warrants a detailed method of study, so that samples are taken at intervals of one foot or less, and the samples and residues are weighed to obtain residue proportions to a fair degree of accuracy.
>
> In our studies to determine the detailed chemical character of the commercial limestones of southern Illinois, the method of insoluble residues has also been used as a guide for delineating lithologic units, where such units are otherwise obscure, as a basis for compounding samples for detailed chemical analyses.

No doubt there are other instances of the direct application of the method, which have not come to the attention of the writer, and it may be repeated that the use of the method will probably become more widespread within a comparatively short period of time.

Suggestions for Using the Method

In any application of the insoluble-residue method it should be kept in mind that such a study must be thoroughly done; that it should follow a definite scheme of organization; that results cannot be obtained when the study is made in haphazard fashion; and that results may not be forthcoming immediately. Intimate features and descriptions of residues that have been published in the reports cited herein, or in others not specifically mentioned, may not always be found in different localities, and the fact that they are absent in any particular locality does not mean that other residual characteristics are not present, or that the method has failed.

It should be borne in mind that in some instances the correlation of formations or stratigraphic units underground with those on the surface cannot in every instance be immediately and successfully obtained, but that time and patience are necessary; also, that certain limestones and dolomites yield small amounts of insoluble material, and that larger samples must be utilized in the preparation of the residues. The absence of insoluble material may in itself be a reliable characteristic, as it is in the particular instance of a formation that occurs in the lower Paleozoic section of Missouri.

In any attempt to utilize this method, it is well to remember that the characteristics of the mass should be kept in mind rather than specific details and variations thereof in any one particular constituent. The writer does not presume to say that the insoluble-residue method is the solution for all the problems, either scientific or economic, that may be presented in an area where limestones or dolomites occur in the geologic section. In some localities, a study of insoluble residues may be of little or no value, but this can be determined only after the method has been applied in a thorough and systematic manner to the problem under consideration.

Summary

This paper describes briefly the insoluble-residue method, the procedure and the history of development. The economic application of the method in Missouri, where it is used in a routine manner, is described in detail, with particular emphasis on its importance in the solution of problems in the field of ground-water geology. Examples of the use of the method of the geology of oil and gas have been cited, and specific details are recorded of its use in mining geology, and in specific investigations of limestones.

Suggestions have been given relative to the use of the method, and the prediction made that its use, even now widespread and in many fields of applied geology, will be expanded greatly within a comparatively short period of time.

Acknowledgments

The writer is indebted to Dr. H. A. Buehler, State Geologist of Missouri, for unfailing counsel, guidance, and generous assistance in studies by the insoluble-residue method, and for constructive criticism of the present manuscript. The writer is also indebted to many geologists for statements regarding the use of the method, some of which appear in this report and are specifically acknowledged.

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DISCUSSION

(*H. A. Buehler presiding*)

M. H. Newman,* Mascot, Tenn.—We have been using this insoluble method in our zinc prospecting in East Tennessee for several years. The scope of the work of these men from Missouri might be measured by states—ours is measured by acres; where they deal with formations, we deal with individual beds.

Our particular problem has to do with the Knox dolomite, which is a heavily bedded dolomite, or magnesium limestone, with some interspersed beds of straight limestone. We always have found difficulty in making close determinations of horizons from raw churn-drill cuttings in this formation. By the use of insolubles, however, it has been possible to locate certain stratigraphic markers within the formation and these markers and the intervals between them are relied upon to identify horizons. For example, in the drilling of a hole a bed is recognized; continuing, the drill passes through a horizon where another marker is expected, but not found; continuing further, a third marker is found. The presence of a pair of such markers plus an interval characteristic of the locality identifies the horizon. Altogether, we have found the insoluble-residue method a most efficient tool for that purpose in that locality.

W. F. Pond,† Nashville, Tenn.—We had a residue project in Tennessee on the Chester formations of the Mississippian, by H. G. Martin, who had previously examined residues from the Chester in Indiana and Kentucky. He found excellent correlations in Indiana, Kentucky and Tennessee, even to the southern part of the state.

H. S. McQueen.—I placed upon the screen a slide showing cruciform sponge spicules and promised to tell something about them. I refer particularly to the regional distribution of these particular sponge spicules, which occur only in the uppermost portions of thick sections of Upper Cambrian Age. I have seen them from deep wells in southwestern Wisconsin and adjacent parts of Iowa and Illinois; we find them in Missouri; they occur also in Oklahoma and in the Upper Cambrian of the central mineral region of Texas.

The striking thing about them is the fact that they indicate always a thick section of Upper Cambrian. In Missouri, at the base of the Upper Cambrian there is a thick sandstone, which generally is an important water-producer. In other words, if these

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* Geologist, American Zinc, Lead and Smelting Co.

† State Geologist of Tennessee.

sponge spicules appear in the residues, we know that if we have made a previous estimate and counted on a thin section of the Upper Cambrian, we had better sharpen our pencils and revise our original estimate.

Sections were carefully measured in the Central Basin in Texas by the late C. L. Dake and field specimens were sent to me for study. Later, Dr. Bridge was in the area with Dr. Dake and collected fossils from the previously measured sections. On the basis of the fossils contained in the beds, they made certain classifications of the section. We prepared residues and I can say, without contradiction and simply to confirm what I have told you here, that the regional distribution of the zones established by the insoluble-residue method and by paleontology corresponded very, very closely.

From a study of insoluble residues, we know that a systemic classification is distinctly in the realm of the possible, for we know that the residues from the rocks of each system have in a general way, characteristics that differentiate them from the rocks of other systems.

H. A. Buehler,* Rolla, Mo.—Five or six years ago, we were not able to get the drillers interested in the work of the geological survey in Missouri. We had to do too much guessing in the difficult Ozark section. Very frequently, it was impossible to give them any real information. Today there is not a driller of importance, a man who does any considerable amount of drilling in Missouri, who does not drive clear across the state to get the answer when he gets into difficulty. McQueen is a little modest. They now swear by his results. Time after time, he has done the same thing as has been done on the Eagle Picher well. I believe the method has application in any section of carbonate rocks.

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* State Geologist of Missouri.

The Corocoro Copper District of Bolivia

By Adrien Berton,* Member A.I.M.E.

(New York Meeting, February, 1936)

For nearly a century, the Corocoro deposit has been renowned among geologists from the fact that it shares with the Lake Superior deposits of the United States the distinction of being the only important copper districts where native copper is worked commercially.

The Corocoro district, named after the city of Corocoro, the capital of the Province of Pacajes (Department of La Paz, Bolivia) is situated towards the western edge of the Bolivian high plateau, at an altitude of a little over 13,000 ft. The most interesting part of the district is that immediately surrounding the city of Corocoro, at a distance, by railway, of 65 miles from La Paz and 210 miles to the Chilean port of Arica on the Pacific Coast. The climate of the region is that of the semi-desert, dry and relatively cold, with big differences of temperature between day and night. The rainy season begins in December and ends in March.

The cupriferous lodes of Corocoro have not attracted as much interest as the silver mines of Potosi but they have been known for a long time. In the precolonial period the Indians were using the *charquis*, or natural native copper sheets, to make small brass objects, and the beautiful pale green brochantite of the outcrops as a source of green pigments.

During the colonial period, the oxidized ores of the outcrops were worked for copper, which was used by the mint at Potosi for making currency. The native copper ores were neglected because the Spanish miners were at a loss how to concentrate and smelt them. About 1830, work by rudimentary processes, which were improved little by little, was begun on the native copper ores. The total production of native copper in the following hundred years amounted to 100,000 tons. This native copper, which was remarkably pure, was much appreciated on the European markets before the introduction of electrolytic copper. It was useful for making special ornamental brass or bronze and the granular concentrates of native copper, called *barrilla*, containing 80 to 85 per cent copper were suitable for making copper sulfate by direct oxidation and subsequent leaching with sulfuric acid.

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* Ingénieur des Arts et Manufactures, Ville d'Avray, Seine et Oise, France.

The completion of the Arica-La Paz Railway in 1912, which took place a few years after the amalgamation of the mining companies, marked the beginning of a new era in the development of the district. This railway permitted the working of the chalcocite ores, called *yanabarra* (from the Quechua *yana*, meaning black, and *barra*, ore). The high-grade sulfide ores, averaging 20 per cent Cu, were exported directly in bags after hand picking. The low-grade sulfides, averaging 3 to 6 per cent Cu, were concentrated by flotation, giving a concentrate of 40 to 45 per cent Cu. The equivalent of more than 100,000 tons of metallic copper has been exported in sulfide form.

The recent universal economic depression caused the shutting down of operations. Before the Corocoro district undergoes the unavoidable fate of so many mines, the author desires to set down a résumé of observations made during the long period of 21 years spent in Corocoro as mining engineer and general manager of the Corocoro United Copper Mines Ltd. As stated at the beginning, the unique geologic position of the Corocoro deposits attracted the interest of geologists; among the numerous papers on Corocoro the principal is that by J. T. Singewald, Jr. and E. W. Berry[1].

Geology

The Corocoro district, in the vicinity of the town, may be briefly described as follows: two series of sedimentary rocks, mainly made up of a thick sequence of sandstones, conglomerates and shales, having a prevailing ferruginous red color, dip respectively westward and eastward from a nearly vertical fault having a strike N. 30° W. (Figs. 1 and 2).

The westerly dipping beds, called "Vetas," having a texture and a color slightly different from those of the easterly dipping beds called "Ramos," which are more ferruginous than the Vetas. On account of these differences between the two series and on account of their unconformity near the fault, it was believed that these series were different and were of quite distinct geological ages. Singewald and Berry (ref. 1, p. 39) have shown clearly, by the examination of fossil plants discovered in the Vetas, that the Vetas are of Pliocene age. The Vetas, aside from their texture and color, seemed to differ from the Ramos by the presence of fossil plants; whereas the Ramos might be identified by the lack of plants and the presence of fossil animals. This discrimination is a mere guess. A close examination of the rocks reveals the fact that both fossil plants and fossil animals are present in the two series. These fossils (plants or animals) are almost always cupriferous. The examination in the field leads us to believe that the Vetas and the Ramos are both of Pliocene age; the Ramos, being superior beds, are consequently younger than the Vetas.

[1] References are at the end of the paper.

Fig. 1.—Corocoro district, looking north. Vetas series on west side of valley and Ramos series on east side. Dotted line shows fault.

Fig. 3 is a geological sketch map of the Corocoro district, made by the author, which covers an area of nearly 2000 sq. km. over which the author walked or drove.

The sedimentary rocks have been folded into anticlines, striking N. 30° W. The principal one corresponds to the fault of Corocoro. The

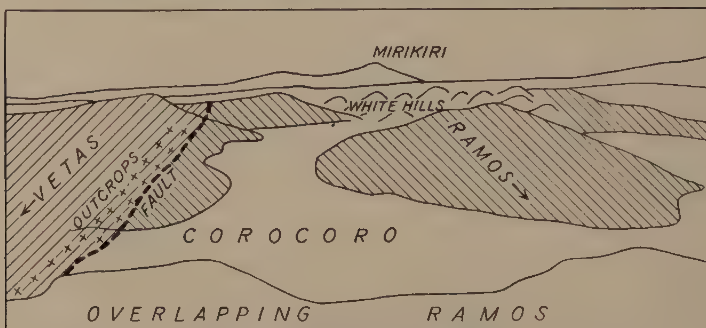

Fig. 2.—Corocoro district.

crest of these anticlines has been razed by erosion which, in places, has removed most of the softer Ramos. Along the fault plane, the Ramos went down in comparison to the Vetas, bringing these two series in

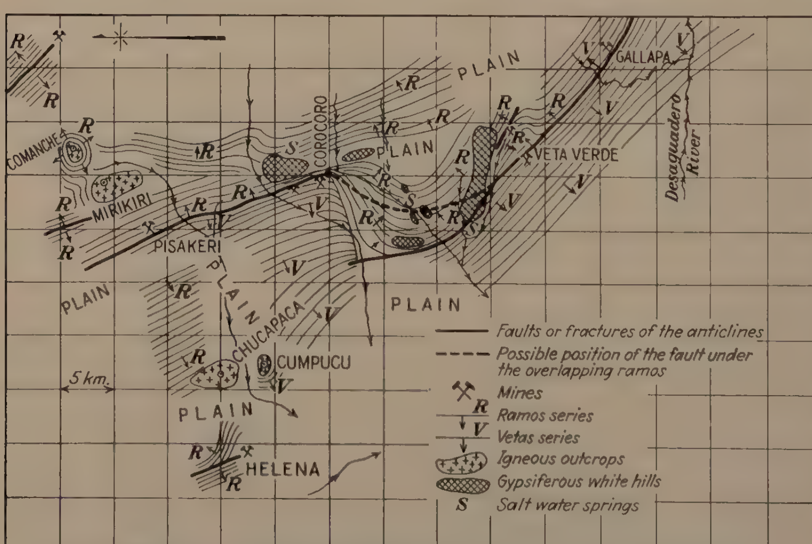

Fig. 3.—Geological map of Corocoro district.

contact. A few hundred meters to the south of the town of Corocoro, in consequence of transverse pressures, directed east to west, the more plastic Ramos have been pushed over the Vetas, superposing themselves on the Vetas and changing the westerly dip of the upper part of the

latter into an easterly dip. This overlapping is the manifestation of a *nappe de charriage*, which extends between 6 and 7 km. west of the fault.

This *nappe de charriage* has been explored not only on the surface but in the underground workings where there is an argillaceous friction zone between the overlapping Ramos and the underlying Vetas. This *nappe* and its argillaceous contact zone have had an important influence on the general direction and distribution of the mineralization in the copper-bearing beds. The recognition of this *nappe de charriage* invalidates beyond question the existence of the Desaguadero series of Singewald and Berry (ref. 1, pp. 49 to 53).

All geologists that have been in Corocoro have noted the outcrops of igneous rocks that have pierced the sedimentary beds. The principal outcrops are at about 20 km. to the northwest of the town of Corocoro. These igneous rocks, protruding several hundred meters above the ground, are dioritic porphyries (Cumpucu—Chucapaca—Mirikiri—Comanche). They seem to be the external manifestation of an underlying dioritic magma, on which, in depth, the sediments are resting. None of the deepest workings in the mines has yet encountered this magma. At a depth of 450 m., the sandstones do not show any metamorphism.

L. Sundt and Singewald, and Berry have observed that certain beds of the Vetas contain angular fragments of these igneous rocks, and that other beds, as well in the Ramos as in the Vetas, are made up of igneous tuffs. Hence, the period of igneous activity has been more or less coincident with that of the deposition of the sediments. This activity was resumed after a pause, when the sedimentary rocks were more or less in their present position. The two outcrops called Chucapaca and Mirikiri form two big mountains with soft slopes and rounded contours, which indicate a very long exposure to the erosion (Fig. 4). On the other hand, the two other outcrops, Comanche and Cumpucu, are peaks with steep, very little eroded slopes; they are of more recent age. The attitude of the sedimentary rocks at the base of the Comanche and of the Cumpucu outcrops shows clearly that they are intrusive; they pierced the strata that have been drawn along upwards (Fig. 5).

Résumé.—All the Corocoro district is made up of sediments, mostly ferruginous and porous sandstones. These sediments have been folded into anticlines with a N.30°W. strike. The Corocoro anticline has been dislocated by faults, which are parallel or perpendicular to its general strike. The sediments rest on an igneous dioritic magma of which the final activity is related to the dislocations of the anticline and also to the subsequent mineralization.

Mineralized Outcrops

A very striking and important phenomenon is that the mineralized outcrops are always in the vicinity of the fracture of the dome of the

FIG. 4.—LOOKING NORTH; IN THE BACKGROUND IS THE IGNEOUS CHUCAPACA, IN THE FOREGROUND THE VETAS SERIES.

The soft contours of Chucapaca indicate long exposure to eroding agents.

FIG. 5.—LOOKING NORTH, SHOWING ERUPTIVE CUMPUCU PROTRUDING IN PLAIN COVERING THE ERODED VETAS.

Abrupt contours indicate relatively short exposure to erosion. *A-B* is a stratum belonging to Vetas and lifted up by eruption of Cumpucu.

anticlines. Copper occurs wherever there is a sharp and consequently deep and dislocated fold. This phenomenon is general in the district. The remnants of fossil organic matter (plants, wood and bones) in the Ramos and the Vetas are frequently slightly mineralized, but they must not be confused with the outcrops and will be considered farther on.

The copper contained in the outcrops occurs in numerous forms: native copper, cuprite, azurite, malachite, brochantite and chrysocolla. The outcrops contain also manganese oxide and in some places remnants of gossan with hematite. At Pisakeri (15 km. north of Corocoro) a

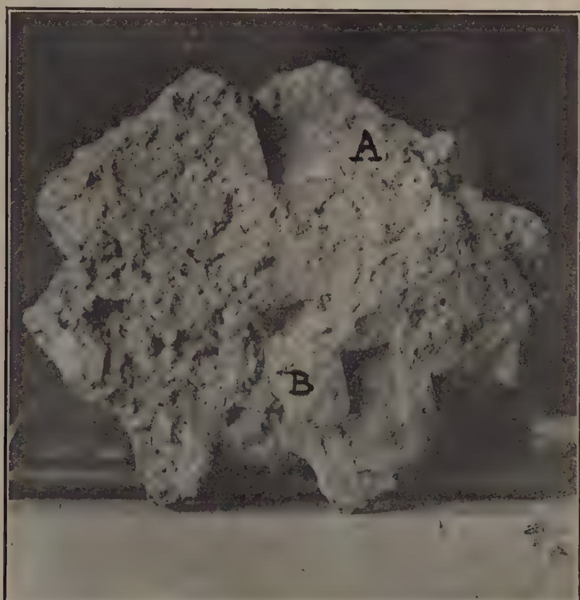

Fig. 6.—Specimen of hematite found in gossan of Pisakeri mine. Shows hollow prints of pyrite crystals, *A* and *B*.

true gossan is found; it possesses the characteristics of pyrite and chalcopyrite weathering. The rock of the gossan is cellular and porous, like a sponge; the cavities are lined with compact or pulverulent hematite with some opalescent chalcedony. A specimen of hematite was found with hollow prints of cubic pyrite crystals, which have disappeared (Fig. 6). The dimensions of these crystals is about 10 mm. In the underlying argillaceous bed, nodules of nearly pure hematite are found. A careful examination of the materials of other outcrops disclosed very minute remnants of gossan.

The study of the district shows very clearly that it has been strongly eroded. The erosion has removed not only the main part of the gossans but also much of the underlying outcrops. Observations, for which there

is no space in this paper, demonstrate that about 30 per cent of the original deposit has been removed by erosion. This erosion has produced large alluvial plains, which perhaps cover other mineralized outcrops. A striking example is that of the Helena mine, northwest of Corocoro, in the middle of a plain, of which the outcrops are covered except in small ravines washed out by rains.

Not very far from the outcrops are white hills (*cerros blancos*). They have been described by Singewald and Berry (ref. 1, p. 34). Their whitish color is due to abundance of gypsum in the rocks, which seem to have grown by accretion, giving a superficial capping that rests on the

FIG. 7.—TYPICAL VIEW SHOWING GYPSIFEROUS WHITE HILLS COVERING UNDISTURBED STRATA OF UNDERLYING RAMOS.

undisturbed underlying strata (Fig. 7). From these white hills issue springs of strong salt water. These springs are utilized by the Indians for salt. In certain places, the white hills are cut by eruptive dikes.

Résumé.—The mineralized outcrops are intimately associated with the fractures of the anticlines. They contain remnants of gossan. They contain copper in all forms of oxidation. They are located in the vicinity of gypsiferous white hills.

UNDERGROUND MINERALIZATION

In depth, the mineralization occurs in the strata on each side of the contact between the Ramos and the Vetas. Copper is found in both the Ramos and the Vetas. The mineralized strata, of which the thickness

ranges from a few centimeters to 4 m. or more, are separated from each other by barren beds. Generally, the sandstone beds are separated from each other by a thin layer of impervious clay. The ores are not restricted to one or more definite horizons, nor are they restricted to any particular facies of the sediments. The distribution of the mineralized strata in the series of beds as well as the distribution of the mineralization in a definite stratum is somewhat irregular.

All the ores of Corocoro, no matter whether they are native, oxidized, carbonated or sulfide, occur in the form of a cupriferous impregnation, which has replaced the original cement of the mineralized sandstones (Fig. 8).

To investigate the distribution of the mineralization in the rocks, the electrical conductivity of several specimens of native copper ores has been tested. The apparatus was very simple, consisting of an electric battery in circuit with a small electric bulb and the specimen. Whenever the circuit was closed on two grains of metallic copper, the bulb lighted, even when the two grains of copper were situated in mineralized spots separated by a barren zone (Fig. 9). Of course the bulb did not light when one of the electric contacts was on a grain of sand. Therefore, in a block of mineralized sandstone, the mineralization is not discontinuous but perfectly uninterrupted; all the grains of copper are intimately bound to each other. The mineralization has spread from place to place, by degrees and continuously, as if under the action of a pressure that forced the copper-bearing solutions through the pores of the sandstones or through the minute cracks of the rocks. The microscopic examination of polished specimens and thin sections of the ores confirms the preceding hypothesis (Figs. 8 and 9). This hypothesis is also confirmed by the occurrence of large sheets of copper filling the fissures in the rocks. The thickness of these sheets ranges between 0.1 and 2 mm. and the surface is as large as several square meters; arborescent forms of native copper are also encountered.

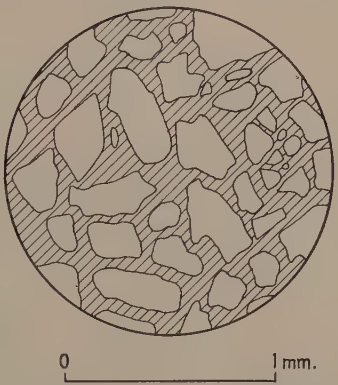

FIG. 8.—TYPICAL SPECIMEN OF MINERALIZED SANDSTONE (CUPRITE AFTER NATIVE COPPER). CAMERA LUCIDA DRAWING.
White shows grains of sandstone, cross-hatching indicates copper.

In the mines of Corocoro, going from the greatest depth ever reached (450 m.) up to the surface, there is a systematic change. First comes a mineralized native copper zone, in red and generally very fine-grained sandstones belonging to the Ramos series. These sandstones are bleached over a certain extent around the native copper. The mineralized spots offer a certain cohesion due to the native copper in the interstices between

the grains of quartz or sand. The bleached part is generally friable in consequence of the removal of the ferruginous cement that was holding the grains; then, the copper grains show a clean, copper red surface, without any mark of oxidation or sulfidation. No sulfide has been encountered in the Ramos. The mineralized Ramos abut against the Vetas series, from which they are separated by the plane of the big longitudinal fault. The mineralization seems to have crossed the fault, spreading into the Vetas, which are more porous and less ferruginous than the Ramos. The filling of the fault has been strongly mineralized, giving a very rich vein, the Dorado (Figs. 10 to 12).

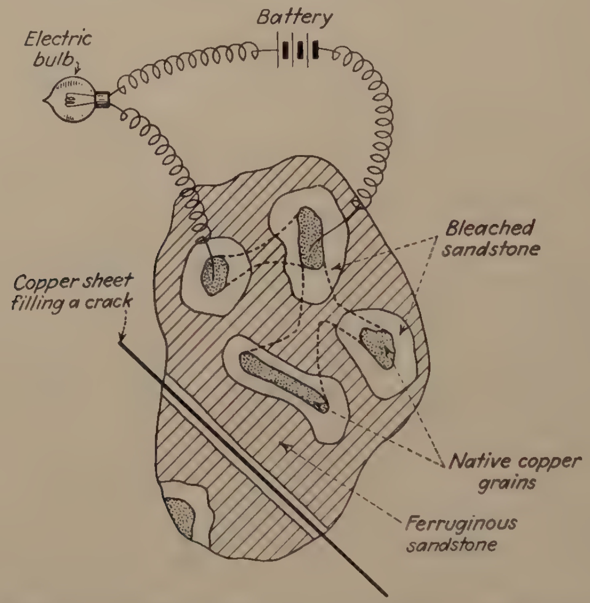

FIG. 9.—SPECIMEN OF SPOTTED MINERALIZED SANDSTONE, SHOWING BLEACHED HALO AROUND COPPER GRAINS.

Specimen has been tested for electrical conductivity. This figure shows that two apparently separate mineralized spots are, in fact, communicating inside the specimen.

In the series of the Vetas native copper beds are also encountered. Other rich lodes in the Vetas contain chalcocite, which at some places is crossed by veinlets of native copper. Under the microscope, a polished section of these veinlets shows clearly that native copper has replaced the chalcocite (Fig. 13). Big masses of metallic copper, weighing several quintals, have been also found. Under the microscope it shows minute specks of unreplaced chalcocite, which tends to prove the existence of secondary native copper after chalcocite.

Nearer the surface, chalcocite is sometimes mixed with blue covellite. In the outcrops pale green brochantite is found. Some specimens contain both chalcocite and brochantite; others show a nucleus of chalcocite and

covellite surrounded by a halo of brochantite. In the chalcocite-impregnated sandstones, cracks are sometimes lined with magnificent

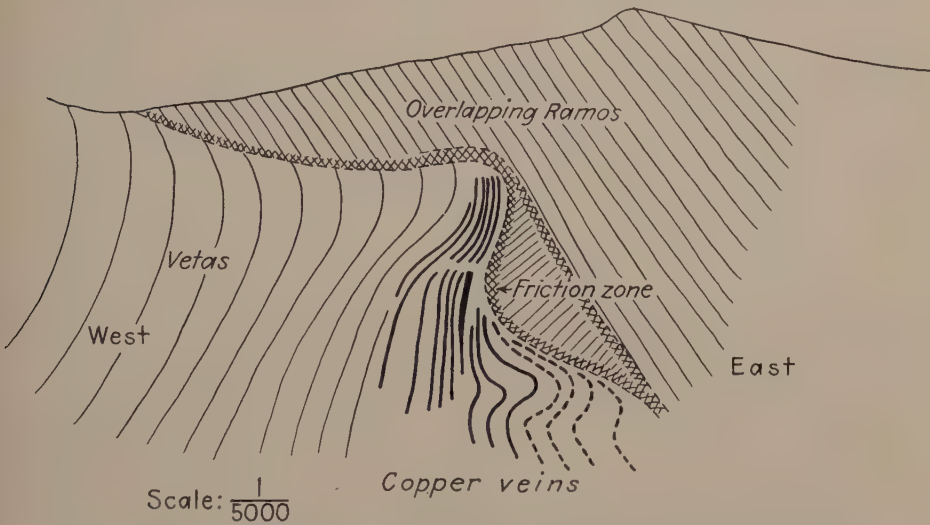

FIG. 10.—CROSS SECTION OF GUALLATIRI MINE, 500 METERS SOUTH OF TOWN OF COROCORO.

Section shows beginning of overlap of Ramos and change of dip of Vetas top. Heavy lines are copper veins.

crystals of cuprite. The sandstones containing chalcocite do not show the ferruginous characteristics of the native copper-bearing sandstones. In the bleached sandstones of the Vetas outcrops, cuprite containing

FIG. 11.—CROSS SECTION OF VIZCACHANI MINE, TOWN OF COROCORO.
Heavy lines are copper veins. Mineralized Ramos do not outcrop.

minute grains of native copper is found. Azurite, malachite, chrysocolla and specks of manganese oxide are encountered. In certain native copper

beds of the Vetas, the metallic copper does not show a brilliant clean surface; the copper grains seem to be painted with a thin layer of chalcocite crystals.

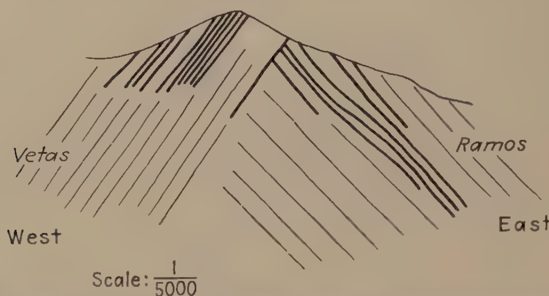

Fig. 12.—Cross section of San Agustin mine, 1700 meters north of town of Corocoro.
Mineralized Ramos are outcropping. Heavy lines are copper veins.

Other ore minerals of minor importance are: native silver, domeykite, and slightly cupriferous and argentiferous galena. All the ores of Corocoro are slightly argentiferous, the tenor in silver ranging from 3 to 6 oz. per ton of copper.

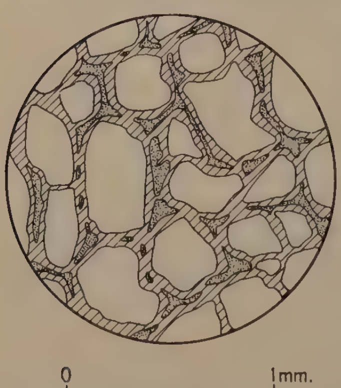

Fig. 13.—Specimen of chalcocite ore in sandstone, showing native copper replacing chalcocite. Camera lucida drawing.
White, grains of sandstone; stippling, unreplaced chalcocite; cross-hatching, native copper replacing chalcocite.

Hexagonal prisms of native copper, pseudomorphic after aragonite, are found in all stages of replacement from the pure aragonite to almost pure copper. Bluish crystals of celestite and druses lined with beautiful brownish crystals of barite have been found. Gypsum is encountered as fillings in numerous fissures in the rocks. The underground waters in the mines are strongly salt and slightly acid. The occurrence of totally bleached sandstones is noteworthy. They are very friable on account of the removal of the matrix. These beds are called *panizo*, and are almost entirely barren.

Interpretation

It has been pointed out that Corocoro has produced about 100,000 metric tons of copper in native copper ores and 100,000 metric tons of copper, mainly in sulfide ores and in oxides. Thus, the sulfide minerals have been equal in importance to the native copper. Let us try to interpret this mineralization to explain the genesis of the Corocoro copper deposits.

It seems reasonable to ascribe an epigenetic origin to the Corocoro deposits and a magmatic deep-seated source to the mineralizers. The abundance of sulfides and the remnants of gossan show that these mineralizers contained copper sulfides in solution, probably chalcocite. It is of interest to note what has happened along the path of the mineralizers from the very bottom of the deposit up to the surface.

In depth, sulfide-bearing solutions spread through the ferruginous sandstones of the Ramos. The ferric oxide of the matrix reacted upon the chalcocite, giving native copper. The withdrawal of ferric oxide would explain the bleaching of the rock around the copper grains. The reactions took place wherever ferric oxide was available, as in the Ramos, in the fault filling, and in certain Vetas. There, the resulting native copper is perfectly clean and rosy. In other Vetas, where ferric oxide was insufficient, the cupriferous solutions have deposited part of their copper content in the native form, proportionally to the ferric oxide contained in the rocks. Once this ferric oxide was exhausted, the solutions were unable to deposit metallic copper. The remaining copper was deposited as sulfide on the already precipitated grains of copper, giving them a blackish appearance.

In other beds, quite devoid of ferric oxide, the chalcocite simply precipitated. In certain places, the chalcocite solutions reached the very surface. The remnants of gossan suggest that an abundant deposit of pyrite and chalcopyrite was formed on the outcrops. The reactions between the chalcocite and the ferric oxide yielded acid solutions, which dissolved the matrix of the bleached sandstones called *panizo*. Such is the possible genesis of the primary Corocoro deposit.

Corocoro has been considered by some geologists as belonging to a very peculiar type of copper deposit, which could not be classified in any known category, and that for two reasons: (1) the cupriferous precipitation in red sandstones; (2) the abundant occurrence of native copper. It is demonstrated that the ascending solutions that formed the Corocoro deposit were capable of precipitating native copper and copper sulfide. The precipitation of the native copper may be ascribed with reasonable certainty to the presence of ferric oxide in the path of the cupriferous solutions through the red sandstones. If these sandstones had not contained ferric oxide, the Corocoro deposit would have been a mere chalcocite deposit. On the other hand, with an abundance of ferric oxide, the deposit would have contained only native copper from the bottom up to the outcrops.

The fact that only certain beds have been mineralized, whereas others in the vicinity have not, may be explained by the local stratigraphy and the dislocations that have directed the solutions. The latter may have been sealed by their relative imperviousness or by thoroughly impervious local faults.

Certain strata, in both the Vetas and the Ramos, situated at a relatively great distance on each side of the ore outcrops, contain slightly cupriferous fossil organic matter. These cupriferous precipitates, generally carbonates, look like a halo around a nucleus of organic matter and are without economic importance. They may be ascribed to more efficient precipitation by organic matter from very weak or almost exhausted copper-bearing solutions that have deposited most of their copper burden in depth through reaction with the mineral reagents. This is happening today in the Corocoro River. There, the rocks, subjected to the action of the slightly cupriferous waters of the mines and of the concentrating mills, do not show any copper impregnation whereas organic matter like wood, bones and rawhide are rapidly impregnated with copper carbonates.

Weathering

The primary Corocoro deposit has undergone oxidation, erosion and weathering. Erosion has been very important because it has facilitated deep weathering. The porous structure of the mineralized rocks, their friability, the alternation of the rainy and dry seasons as well as the big differences between the diurnal and nocturnal temperatures, corresponding to the semi-arid climate of the region, have made the oxidation easy. On the surface, there was a large quantity of pyrite and chalcopyrite, which, by oxidation, gave solutions loaded with sulfuric acid, ferric sulfate and copper sulfate. The copper sulfate, reacting on fresh pyrite, has produced secondary chalcocite and covellite. The covellite has been transformed into chalcocite. In other parts, where pyrite was less abundant, secondary native copper precipitated after reaction between chalcocite and ferric sulfate. Thus the descending solutions have produced a zone of secondary enrichment.

The primary or secondary native copper has been totally or partially oxidized to cuprite.

Chalcocite, by oxidation, has also given cuprite. The chalcocite and the covellite have been transformed also into brochantite.

The atmospheric carbon dioxide has produced copper carbonates like azurite and malachite.

Summary

Corocoro may be classified as an epigenetic copper deposit and the mineralizing solutions may be ascribed with reasonable certainty to a magmatic, deep-seated source. The mineralizers have precipitated primary native copper or primary chalcocite, according to the chemical nature of the matrix of the impregnated sandstones.

Subsequent weathering has produced secondary enrichment of the upper parts of the deposit with secondary native copper and secondary chalcocite.

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DISCUSSION

(*Edward Sampson presiding*)

J. T. Singewald, Jr.,* Baltimore, Md. (written discussion).—It is right that Mr. Berton should be included in the long list of contributors to the geology of Corocoro. For many years he accorded facilities to visiting geologists and gave them the benefit of his observations. Much that has been published on this district by others is in part at least to be credited to him.

For the most part this paper is a concise summary of our knowledge of the geology of the district, which has been published elsewhere and concerning much of which there is no disagreement. Mr. Berton presents only the conclusions with which he is in accord. He does not go into the details of the evidence upon which some of these conclusions are based, nor does he discuss views that are at variance with them and the arguments in their support. For such information the student of Corocoro must turn to the literature.

In 1922, Singewald and Berry summarized the literature (ref. 1) to that time and presented in detail the combined results of Singewald and Miller's visit in 1915 and that of Singewald and Berry in 1919. Since that time a number of other papers on Corocoro have appeared, largely in German and in part rather inaccessible. The latter have received no attention in Berton's paper.

A completely new contribution by Berton is the demonstration of the permeability of the ore beds by their electrical conductivity. His explanation of the formation of secondary chalcocite has not been proposed before for this district.

Age of Vetas and Ramos

Berton states that the age of these beds is Pliocene, without presenting the evidence. In 1915 he and Dorion called the attention of Singewald and Miller to the occurrence of fossil plants in the Vetas. From their collections, later corroborated by additional collections by Berry and Singewald, Berry determined the age of the Vetas as Pliocene. That the Vetas are as young as Pliocene has been questioned by Steinmann and Brüggen. In 1922, Steinmann[6] inclined to the view that they may be as old as upper Oligocene or middle Miocene. The basis of this opinion was that there is no absolute measure of the rate of change of floras in tropical latitudes under

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[6] Steinmann: *Geologische Rundschau,* **13,** 1–8.

uniform climatic conditions, so that age determinations inevitably will be subject to certain variation according to the scale or rate of change assumed. A more positive divergent opinion and a still older Tertiary age was expressed by Brüggen in 1933. He questions the Pliocene age on the same grounds as Steinmann; namely, that the method of determining age on the basis of percentage of living forms used for marine faunas may not be applicable to Tertiary tropical faunas. Further, the geologic events that transpired subsequently to the deposition of the Vetas are too numerous and great to be compressed into the relatively short subsequent time. He says a similar problem was presented by the Concepcion beds in Chile. The flora appeared very young Tertiary, yet fossiliferous marine intercalations proved the Eocene age of the rocks. Brüggen consequently considers the Vetas to be of Eocene age also. The Eocene age of the Concepcion beds is not accepted without question, they have been regarded as young as Miocene; and the Corocoro beds are not necessarily as old as they.

Though there may be basis for differences of opinion as to the exact correlation of the Vetas, there is no question of their Tertiary age. This has been established since 1917. Yet in the first three editions of his "Mineral Deposits," 1913 to 1928, Lindgren gives their age as Permian. In the fourth edition, 1933, he still ignores the paleontologic evidence, and adopts Steinmann's old lithologic correlation of 1906 as Cretaceous, which in the meantime Steinmann himself had abandoned and which Lindgren had hitherto not accepted.

Desaguadero Series

On Cerro Corocoro, the Vetas and Ramos, the Corocoro fault between them, and the ore deposits are covered unconformably by a group of red beds. Singewald and Berry considered these a younger overlying formation which they called the Desaguadero series. Brüggen in 1934 described them (ref. 5) as an overthrust *nappe* of Ramos. Berton also considers them a *nappe de charriage* of Ramos overthrust from the east. Both describe an argillaceous and gypsiferous friction breccia at the base of these beds. Berton's cross section shows a simple overthrust of the Ramos with a local reversal of the westerly dip of the Vetas near the overthrust plane. Brüggen gives a much more detailed and complex picture. His cross section through Cerro Corocoro shows the *nappe* consisting of two overturned folds, one thrust over the other, with their undersides squeezed out. On his geologic map as the Vetas approach the front of the overthrust they not only reverse their dip but in a distance of less than 500 meters undergo a right-angle change in strike by the southerly strike swinging to the west. In the overthrust sheet itself, southwest of Cerro Corocoro, the Ramos undergo an abrupt right-angle change in strike from northeast to nearly west. These northwesterly striking Ramos are separated from the northeasterly striking Ramos on Cerro Corocoro by a strip of the friction breccia only 200 meters wide.

On their geologic map, Singewald and Berry show the contact between the Vetas and the Ramos southwest of Cerro Corocoro as a fault. This fault is in the position of Brüggen's overthrust fault. They note the reversal in dip of the Vetas as they approach the fault. They also describe considerable variations in strike and dip in their Desaguadero beds on Cerro Corocoro, indicating structural irregularities. Hence it is quite likely that Brüggen and Berton are correct in considering them overthrust Ramos beds and not a younger overlapping series of beds. Whether the structure within the overthrust sheet is as complex as described by Brüggen is open to question. A simpler structure, more like that shown by Berton, is perhaps more probable.

Origin of the Ores

In his explanation of the origin of the ores Berton adheres to the theory first advanced by Steinmann in 1906 to explain the native copper and subsequently elabo-

rated by Berry, Miller and Singewald to include the large bodies of chalcocite ore that had been developed since 1912. The same explanation has since also been applied to the Lake Superior copper deposits. Undoubtedly it is the only satisfactory theory that has been advanced.

In 1928 Geier (ref. 2) proposed a theory similar to some of the earlier explanations. He agreed that the mineralizing solutions came from an underlying magmatic source, but he postulated acid thermal waters carrying copper as chloride or bicarbonate and containing small amounts of sulphureted hydrogen. The acid solutions dissolved the iron oxides of the rock and bleached it, and native copper was precipitated through the absorptive action of the kaolinic material in the rock. A widely different theory was proposed by Ahlfeld in 1933 (ref. 4). He did not consider the deposits of hydrothermal origin but believed the ore beds were first impregnated by descending cupriferous solutions that had collected in arid basins, and that this original mineralization was later further concentrated by descending cold waters. The copper was supposedly derived from the weathering of the volcanic rocks of the western Andes and existed in the solutions as chloride and sulphate. Native copper was deposited through the reaction of the copper sulphate with ferrous sulphate. He does not explain the presence of ferrous sulphate under such conditions. Ahlfeld has recently revisited Corocoro and has written me that he has abandoned this theory and now accepts the hydrothermal origin of the deposits.

Secondary Enrichment

The second wholly new contribution by Berton are his ideas on secondary enrichment at Corocoro. At Pisakeri, 15 km. north of Corocoro, he found a piece of hematite gossan with hollow cubic prints, which he says were once filled with pyrite, and the gossan he says possesses the characteristics of pyrite and chalcopyrite weathering. This apparently is the sole basis of his assumption that at Corocoro at the surface the ore deposits contained a large quantity of pyrite and chalcopyrite and that supergene chalcocite was precipitated through the reaction of copper sulphate solutions derived from the chalcopyrite with the pyrite. Certain it is that one of the characteristics of Corocoro ores is the absence of iron sulphides and cupriferous iron sulphides now. That has been recognized by everyone. If Berton's explanation is correct we have the remarkable coincidences that supergene enrichment has penetrated exactly to the bottom of the hypogene zone of pyrite and chalcopyrite and that it has been complete throughout that zone without leaving any remnants of the hypogene iron sulphides from which the supergene chalcocite was deposited. Moreover, Berton describes the deposition of chalcocite in his hypogene mineralization. He has, as it were, a hypogene zoning with a deeper chalcocite zone capped by a higher pyrite-chalcopyrite zone, also a very unusual phenomenon. In view of the complete absence of cupriferous iron sulphides, geologists will hardly admit the former existence of such zoning. Without the zoning there is no basis for the supergene chalcocitization which he describes.

The explanation advanced for the origin of the ores is adequate to explain both the native copper and the chalcocite, without the necessity of supergene enrichment. This is corroborated by the Lake Superior district. It must be admitted, however, that the distribution of the sulphides at Corocoro is troublesome. The sulphides are restricted to the Vetas and to the upper part of the Vetas. Below them is native copper. The copper content of the sulphide ores is also several times as great as that of the native copper ores. These facts are very suggestive of supergene chalcocite enrichment. On the other hand the hypogene mineralization is of a character not favorable to chalcocitization. Doubtless it was because of this difficulty that Berton assumed a surface zone of pyrite and chalcopyrite.

Geier and Brüggen, the only other writers that have described supergene chalcocite at Corocoro, have also run into difficulties and inconsistencies. Geier writes that the

occurrence of primary chalcocite is probable and that of secondary chalcocite is certain. The uniformly disseminated chalcocite in the upper levels he thinks may be primary, but the richer accumulations he considers secondary. The massive texture developed by etching polished surfaces is the evidence of its secondary nature. In the cementation zone, however, he described only supergene native copper, saying "whether in the formation of the copper instead of iron sulphides which are absent the large quantities of weathering alkali silicates or whether ferrous sulphate caused the precipitation of the copper from copper sulphate is still uncertain." The formation of the secondary chalcocite he places in the zone of oxidation. Its mode of formation is explained in the single sentence: "The richer occurrence of chalcocite in the oxidation zone may be ascribed to the destruction of primary native copper by circulating sulphate waters derived from the leaching of gypsum lenses in the rock." A most extraordinary way of forming chalcocite! Geier's discussion is both unorthodox and unconvincing.

Even less logical is Ahlfeld's discussion. In his primary mineralization he has only native copper, finely and sparingly disseminated as impregnation of the matrix of the beds. With this as a starting point, he has descending meteoric waters develop the following profile:

Oxidation Zone:
- Top: Malachite, azurite, brochantite.
- Middle: Same plus cuprite and secondary native copper.
- Bottom: Secondary native Copper.

Cementation Zone:
- Secondary chalcocite, rarely covellite.

Deep Zone:
- Primary native copper.

Ahlfeld does not seem to have recognized the inconsistencies of such a profile, nor did he feel called upon to explain the chemistry of the chalcocitization. Concerning the chalcocite he merely says that it has all the properties of a typical cementation ore, etching shows it to be the orthorhombic modification, there is only one kind, and that it replaced native copper. Certainly the formation of secondary chalcocite from a primary native copper ore requires more explanation! Also the development of a zone of oxidation with a bottom of native copper and a top with brochantite from a chalcocite cementation zone requires explanation. As stated above, Ahlfeld has recently abandoned his descension theory for the primary mineralization and accepted its hypogene origin. Possibly with it goes also his supergene chalcocite, certainly the way in which he has explained it must be abandoned.

Whether or not the Corocoro deposits have a zone of supergene chalcocite may be an open question. The mineralogy of the primary ores is not such as favors extensive chalcocitization. Neither Berton's, nor Geier's, nor Ahlfeld's evidence for a zone of chalcocitization is satisfactory. It will require more careful study of the chalcocite itself and of its distribution to prove the existence of such a zone. The formation of such a zone must be explained in some way that is more consistent with observable facts, and recognized geologic processes than are any of the explanations thus far offered.

After having been at a complete standstill for several years, the mines are being unwatered and operated again by the American Smelting and Refining Co. Possibly the data that are lacking will be supplied by some member of its staff.

Aerial Geologizing

(New York Meeting, February, 1936)

THE Section on Aerial Geologizing of the American Institute of Mining and Metallurgical Engineers convened on Monday afternoon, Feb. 17, 1936, during the Annual Meeting of the Institute. Mr. Theodore Marvin presided.

CHAIRMAN MARVIN.—We hope to get through with the papers on the program in time so that you can examine the instruments that are displayed here, and also the photographs and air views brought by the Fairchild people.

This is what we might call the first bread and butter session of the Aviation Committee. The group, as many of you know, was started at the request of the Board of Directors two years ago. We have had two sessions since then, both of them of a somewhat general nature, their purpose being to point out the reasons why mining and petroleum engineers should be more interested in the facilities of aviation.

Today we have two papers. The first is one by Mr. Eliel, of the Fairchild Company's California branch. In Mr. Eliel's absence, Mr. Gale, of the Fairchild Company, will read it.

C. H. GALE.—Before I begin to read Mr. Eliel's paper, I would like to point out two instruments that he mentions. One is what he calls the solar-navigator, a new type of aerial navigating device which he has developed for use in flying cameras in aerial surveys. Flying a camera at high altitudes over unfamiliar territory is a very difficult procedure and only a few pilots in this country can do it. So far as I know, this is the first time this device has been discussed in public. Another facility I want to refer to in advance is the use of the stereoplanigraph in making contour maps from photographs. This is an extremely interesting procedure, which we are appreciating more than ever at the present time.

Aerial Reconnaissance and Contour Mapping in Mining

By Leon T. Eliel*

(New York Meeting, February, 1936†)

Ten years ago a broad knowledge of aerial mapping, coupled with a smattering of geology, qualified one to speak on the subject of the application of aerial mapping to geology. Today, with aerial maps a common tool in the hands of expert geologists on every continent throughout the world, it would be nothing short of presumptuous for the author of this paper, who is not a geologist, to give any geological statements on his own authority. Relatively, today we are only purveyors of general geological gossip, with a definite case here and there by way of illustration. But we can present a story of the really marvellous strides that have been made in the evolution of aerial mapping, built around a skeleton of geological tidbits, from which some useful conclusions may be drawn.

The average geologist goes about reading his aerial maps much as one would approach the problem of deciphering a cryptogram. To begin with, typical key expressions appear in recurring sequences and under certain conditions. If once the meaning of these isolated bits of information can be ascertained, the message begins to take meaning. In a similar fashion, the geologist works out his key from aerial maps of territory with which he has some familiarity. He learns that a certain type of thin, white line may mean a vein of quartz. He sees that a certain change in the photographic tone of vegetation on the ground reflects two kinds of soils on opposite sides of a contact. He sees that a certain type of alignment of topographical features often means a fault. He learns that by the erosion pattern he can frequently distinguish soft formations from harder ones. And so, bit by bit, he ties together the information of the picture with his experience on the ground until he develops a reasonably creditable capacity of interpretation.

Through the complete gamut of surface conditions, ranging all the way from flat, arid, exposed desert terrain to dense jungle, or rugged mountains, the evidence from which the geologist must make his interpretation varies with bewildering profusion. The quality of photographs which are obtainable may range from perfect pictures in the desert to almost useless smudges in the steaming jungles. One thing he has learned with certainty—the airplane carrying its mapping camera is a superb reconnaissance tool, only awaiting his ingenuity in correlating the pictures with the ground to simplify most exploratory problems.

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† See page 559.

There are three general classifications covering the application of aerial mapping to mining and oil problems: (1) areal geology; (2) detailed geology; (3) operations and development.

Under the first classification fall such projects as the mapping of great tracts in South Africa, Venezuela and the East Indies. The problem in such work is largely one of elimination. It is a question of selecting certain areas showing some evidence of geological interest.

The second phase of the application of aerial mapping to detailed geology is well illustrated in the intensive use that has been made of pictures in the western United States. Here the individual pictures are carried into the field to assist the geologist in locating himself, to interpret his individual findings, and correlate them with the general surface indications.

The third application of aerial mapping to the operation and construction of properties has many illustrations. Oil companies and mining companies have ordered large-scale aerial photographic maps and contour maps in many parts of the world to assist in the office engineering, preceding the actual construction operations.

Problem Influences Methods

The particular methods of aerial photography that would be employed for an exploratory problem would depend upon both the nature of the terrain being mapped and the character of information sought. If, for example, quartz veins averaging only a foot across are the key to the particular search, the aerial photographs would have to be taken in considerable detail and under conditions of good visibility. If, on the other hand, large anticlines are being hunted, there is no reason why the exploratory mapping could not be done from the ceiling of the airplane with a multiple-lens camera, mapping hundreds of square miles at each exposure. Between these two extremes a wide variety of possibilities occur and each problem should be analyzed individually.

For example: the copper deposits at Bisbee, Ariz., occur in igneous intrusions surrounded by altered sedimentary formations. With such a key the surrounding country might be scoured for somewhat similar conditions. These features are bold and readily seen on pictures of the smallest scale. The Colorado Desert is relatively free of vegetation; conditions of visibility are excellent; and an exploration in this region with this purpose in mind could be effectively conducted at the rate of several thousand square miles in one flying day, using multiple-lens cameras, at a very low cost per square mile.

If time could be rolled back a few decades and this method applied to the quest for oil in California, for example, the small-scale multiple-lens pictures would almost at a glance have centered attention on such outstanding structures as Coalinga, Kettleman Hills, Elk Hills, Belridge,

Elwood, McKittrick, Kern Front, Oak Ridge Uplift, Ventura Avenue, Signal Hill, Baldwin Hills, and several other areas. The pictures would also, of course, have drawn the investigator to a number of very attractive looking dusters. The percentage of success of such a program would have been very favorable in California, where aerial mapping is possibly at its greatest advantage. Conversely, there are great blocks of the Mid-Continent area where the aerial map would have proved a complete failure as a reconnaissance means, and where even today the greatest advantage of the aerial pictures lies in their use for preparing property maps, acquiring right-of-way, and for other similar purposes.

In mapping large areas the ability of the airplane to perform at reasonably high altitudes and to fly accurately the successive strips of overlapping pictures, so that flying is not wasted and so that no holes or gaps are left in the flight, are of paramount importance. Where small-scale pictures will give sufficient detail for geological purposes, the airplane used should be capable of maintaining a sustained altitude of 25,000 ft. above sea level. This means, of course, that the engine must be supercharged and that an auxiliary supply of oxygen must be given to the personnel. The question of the physiological effect of continuous flying at high altitudes upon personnel has been very controversial. The scarcity of oxygen may be readily taken care of, but the effect of the reduced pressure at high altitudes has, to date, placed 25,000 ft. as the safe limit of human endurance for day in and day out operation. Above this altitude the blood no longer absorbs a sufficient supply of oxygen, regardless of the abundance of the supply. These starved corpuscles of the blood die and can never be replaced. Permanent impairment of the health would result from continued flying at high altitudes, and the mapping airplane of the future will unquestionably be supplied with a hermetically sealed and supercharged cabin, maintaining a supply of oxygen under pressure equivalent to an elevation of perhaps 10,000 feet.

One of the most annoying problems involved in the mapping of large areas of which no maps of any sort exist is the navigation of the airplane. Experience has shown that pilots that are considered top-notch by all ordinary criteria are unable to fly strips from 60 to 120 miles long sufficiently parallel to each other so that no gaps are left between the strips, and still hold the overlap down to a reasonably efficient figure. Imagine being in an airplane 20,000 ft. above the ground, over a territory about which nothing is known, trying to keep the airplane going down an imaginary line, constantly changing its heading to compensate for variation in wind direction and velocity.

The one pilot who could measure up to this difficult task, in the writer's entire experience, did it by reason of some unusual instinct, rather than by any reasoning process that he could explain to assist another pilot.

The problem is still further complicated when flying under tropical conditions, where the mantle of haze from the jungle obscures the pilot's view ahead and to the rear and thus deprives him of distant landmarks, which are generally very helpful. This kind of a problem can be visualized by imagining that the flying operation is over the ocean with no land in sight. The monotony of jungle detail is almost as complete as that of water. Of course, aerial navigation over the ocean is being accomplished successfully and the airplane navigator of today generally knows where he is within a few miles. In the problem under consideration, however, a variation of one-half mile to one mile from the theoretical course is the maximum tolerance.

The Solar-Navigator Device

To solve this problem, a navigating device has recently been perfected, designed especially for the mapping airplane. It consists of a drift indicator in the floor of the airplane, stabilized by a gyroscope to offset the tendency of the images to meander over the drift field as the airplane rolls. The drift indicator is suspended from a sun compass in the ceiling of the airplane. The sun compass registers by remote control on a galvanometer on the pilot's instrument panel. The desired course is set between the direction of the drift lines of the drift indicator and the axis of the sun compass. This combination registers on the pilot's instrument panel when he departs in his direction of movement over the ground from the desired direction. It automatically solves the problem of varying winds and drift, and the pilot now merely has to fly his airplane steadily and follow the galvanometer hand on his instrument panel. When it tells him to turn right, he turns right. Aside from this, the operation of the instrument is completely automatic and is only dependent upon careful adjustment by the navigator, back in the cabin, of the drift indicator so that the image of objects on the ground glass move accurately along the drift lines.

With this instrument there is no longer any problem of difficult navigation in connection with aerial mapping, and any pilot that is competent in handling his aircraft can now turn in acceptable results in the cockpit of a mapping plane.

This navigating device, called a "Solar-Navigator," offers additional uses in connection with the reconnaissance of a large, unknown area. The instrument will fly the airplane down any predetermined course to an accuracy of one-quarter degree. With its assistance, the airplane may be started over any known station, such as an astronomical station or a Coast Survey station along the coast line, and from two such known points can carry triangulation far into the interior of the country, taking a series of photographs at all points where each triangulation flight line intersects every other line. In this way a triangulation net may be built

up to an accuracy of one-quarter degree, by means of which, at very little expense and in a very short time, fairly accurate control is established in naccessible regions.

From control established by this means, or any other means, for that matter, a new method has been devised by which it is no longer necessary to depend upon matching images of one picture to the next in order to assemble a mosaic. This latter method has always proved particularly futile in country where there are great differences in the elevation of the terrain. It is because of this factor that, for so many years, aerial maps were always considered to be a very rough and inaccurate method of assembling map information. With the new methods now available for map assembly, reasonably accurate maps can be put together in amazingly short time without anyone setting foot within the territory. Both the navigation instrument and the method of assembly are developments of the last year and afford some index of the amazing progress that is occurring in this art.

Application of Aerial Geology to Mapping

We come now to the question of the application of aerial mapping for detailed geological studies. For such purposes photographs will be taken at scales ranging from 1 in. to 400 ft. up to 1 in. to 2000 ft. In this connection it should be clearly kept in mind that in speaking of aerial maps the scale of the original pictures secured is the governing factor of definition, rather than the altitude of the airplane. By the use of a lens of 12-in. focal length, an airplane flying at 12,000 ft. will ordinarily secure pictures showing as much detail as the photograph taken by a 6-in. lens in an airplane flying at 6000 ft. Therefore, the altitude of the airplane in itself tells nothing about the detail of the picture secured.

The Yellow Aster mine, near Randsburg, Calif., and the vicinity surrounding it, was mapped at a scale of 1 in. to 1000 ft. For the geology prevailing in that region it was found that this scale is very satisfactory. The glory hole at the Yellow Aster mine is revealed by the aerial maps to occur at the intersection of a number of light colored veins with a series of faults. In other words, the glory hole occurs in a zone of considerable fracture along these quartz veins.

Copper areas in Nevada have been variously mapped at scales ranging from 1 in.–1000 ft. to 1 in.–1500 ft. In this region, faults are of extreme importance and it has been found that maps at the smaller scale of 1 in.–1500 ft. pick up the faults very readily in this open region where the ground is unobscured. In one such property a large sum of money had been spent in development work, and proved to be a total loss. Aerial maps showed definitely that the area on which the development money had been spent was isolated from the main body of ore by a fault, which had not been discovered on the ground.

Scales of 1 in.–1500 ft. have proved effective for certain types of gold prospecting where the location of old stream channels is the primary problem. Fragments of old stream channels are frequently picked up by the aerial map in fairly rugged country, with the existence of present-day drainage, hundreds and sometimes even thousands of feet below the old channels. In the Mother Lode country of California the old channels were preserved by lava flows, which can, of course, instantly be picked up by the aerial map.

One of the serious problems in all mining activities in arid regions is the development of an adequate water supply. For this purpose the aerial map has frequently been called upon; as, for example, by the American Potash & Chemical Corporation at Trona, Calif. Aerial maps were made of the mountains surrounding the dry lake bed, on which the mining activities are conducted, and water was secured along some of the fault lines in these mountains. Drainage is an important problem in any operation on a dry lake, and, in working out the engineering details of these drainage problems, the aerial map is particularly helpful, as it indicates every point from which water may flow on to the lake and suggests means of diverting the water into controlled channels.

Aerial maps have been extensively used in investigating and confirming the geology around gold mines in the western United States. These pictures are generally taken at a large scale, and, in many instances, confirm or discredit suspected faults and contacts, and help to trace partly obscured features, such as veins. It is a never-ending source of amazement to people who have not used aerial maps extensively that small-scale pictures can reveal information that cannot be detected from an examination of the ground itself. On careful reflection, the reason becomes apparent. The geologist on the ground sees only what is immediately near him. One isolated bit of evidence often has little significance. However, if that is an outcropping of a vein, which has another little outcrop a half mile away, and another a mile beyond that, this alignment and continuity enables the geologist using the aerial map to correlate his information. This same thing is particularly true of faults and contacts, which frequently show up strikingly in the photograph over long stretches of country. These are often difficult to find on the ground, because of the gradual melting of one formation into the other in such a way that definite evidence is not clearly available.

Many complicated problems of oil geology, such as those existing in Kettleman Hills, Calif., have been solved by the aid of large-scale aerial maps, exactly as the maps have aided the working out of detailed geology in many mining projects. Wherever the geology is intricate and very broken up, the aerial map is generally of material aid, providing the ground can be clearly seen. Sometimes, when the ground is covered by a stand of grass, for example, pictures taken in the spring, just after the

beginning of the growing season, and again in the autumn, will reveal a contact through the color of the foliage, which, by its accelerated growth in spring and its prolonged life in fall, indicates more moisture than the foliage on the opposite side of the contact.

The third application of aerial mapping pertains to the development and operation of properties. Two kinds of maps are available for such purposes. In the first place, a precise aerial photographic mosaic map may be compiled at a large scale, affording a very detailed map of any property, the value of which is self-apparent to any map user.

Preparation of Precise Contour Maps

The second service available to the operating property is the preparation of precise contour maps by the aerial methods. This art is known as "photogrammetry," and, while it has been practiced to some extent for more than 20 years, it is only in the last few years that the method has had general acceptance and usage.

It seems strange, when looking at an aerial picture, that any sort of reliable contour map could be compiled from it. If such a question lurks in your mind, please make it a point to familiarize yourself with the use of the stereoscope. Any two overlapping aerial photographs, taken successively as the airplane flies along, and having a common area, can be viewed in relief by the use of a stereoscope, so that one eye sees one picture and the other eye sees the other. You are getting precisely the same view of the ground that a giant would get were he tall enough to have his two eyes in the successive positions of exposure at which the pictures were taken. Imagine a giant 15,000 ft. tall, with eyes a mile apart! He would get the same sense of relief of the earth itself as you would get looking at a perfect miniature model held only a foot from your eyes! And that is exactly the view which you get when you use a stereoscope. Obviously, there are certain definite geometrical principles which account for this phenomenon of stereoscopic vision. Your eyes follow exactly the same angular paths that two transits would follow were they placed in the position at which each picture was exposed. As the transits would have to converge on a high object, which is closer to them, so your eyes must converge to meld the two images of this high point. When your eyes converge, your brain registers that that object is closer to you. As the transits would diverge, when viewing from the two camera stations to a lower, more distant point, so your two eyes, in following the same light paths to the images on the pictures, must diverge to meld these images. This divergence of the axes of the eyes records on your brain, indicating that the object is further away. The amount of this convergence and divergence is an exact mathematical quantity.

The remarkable stereoscopic plotting machines, such as the "Stereoplanigraph," from which today we plot precise contour maps from aerial

photographs, or photographs taken from the ground, replace the two pictures in miniature, as they were at exposure. The exact angle and elevation of each picture is reproduced at the small scale of the stereoscopic model. Then, while viewing the picture stereoscopically, an index or reference mark is provided, which is so mechanically articulated by the machine that it can only appear to move in a single horizontal plane. This index mark appears to float in the air—if it is out over a canyon—and it appears to bury itself in the plastic ground, providing it is deeper than the ground surface. The operator can tell exactly when the index mark is in precise contact with the surface of the ground. By the operation of two hand wheels, he causes this index mark to trace off exactly the surface of the plastic image, and, because this mark is mechanically restrained so that it can move only in the desired horizontal plane, it is by this operation actually following a contour line, which the machine draws on a piece of paper, forming a map.

Maps can be made by this means to any desired degree of precision. They are made at scales as large as 1 in. to 20 ft., and may be made at scales as small as 1 in. to 2 miles. Under most conditions maps may be made by this method more economically than by conventional ground methods to the same degree of accuracy. Of course, a certain minimum skeleton of surveyed control points on the ground is necessary for the proper preparation of a photogrammetric map. For example: a map at a scale of approximately 1 in. to 1000 ft. requires a known control point for each square mile of territory. Both mining companies and oil companies are today using contour maps developed by photogrammetric methods for the development and operation of their properties.

Costs of Mapping

This is all very interesting, you are probably thinking, but how much does it cost? The cost of any kind of an aerial map, be it a photographic mosaic map or a contour map, is dependent upon many factors. First is the distance away from an operating base. Second is the weather factor. Third is the scale at which the pictures are required. Fourth is the precision at which the map is to be made. And there are many minor factors. It is extremely dangerous to generalize on the matter of the cost of aerial mapping, as there are seldom two projects with sufficiently similar conditions to justify even a rough comparison. Maps that have been made in the past have ranged in price all the way from $2 to $50,000 a square mile. Recently the United States Government has had rather accurate photographic mosaic maps made of about one hundred thousand square miles in southwestern United States, at an average cost of about $4 per square mile. The scale was 2 in. to the mile. Maps made to similar specifications in the East Indies or in the Amazon Valley might cost 10 times this amount. Or, on looser specifications where less

control is required and where high accuracy is not quite so important, the cost might run from $10 to $20 a square mile in the tropics. If the area were 1000 square miles or less, the price per square mile would obviously be much higher, to cover the cost of equipping the expedition and getting it down there. If the area were suitable to multi-lens photography, the cost would be much less. So, on the question of cost, we leave you in almost as much of a quandary as when we first took up this subject, and can only say that it is not a difficult matter to estimate the definite projects when they come up, and that is really the only way to find out how much it is going to cost.

Aerial mapping has developed into a very complex and highly specialized art. There are splendid modern ways of taking shortcuts to results. If you learned something about aerial mapping five or seven years ago, give yourself the privilege of being brought up to date, as five years ago in aerial mapping was about the end of the Cretaceous period.

[For discussion of this paper, see page 579.]

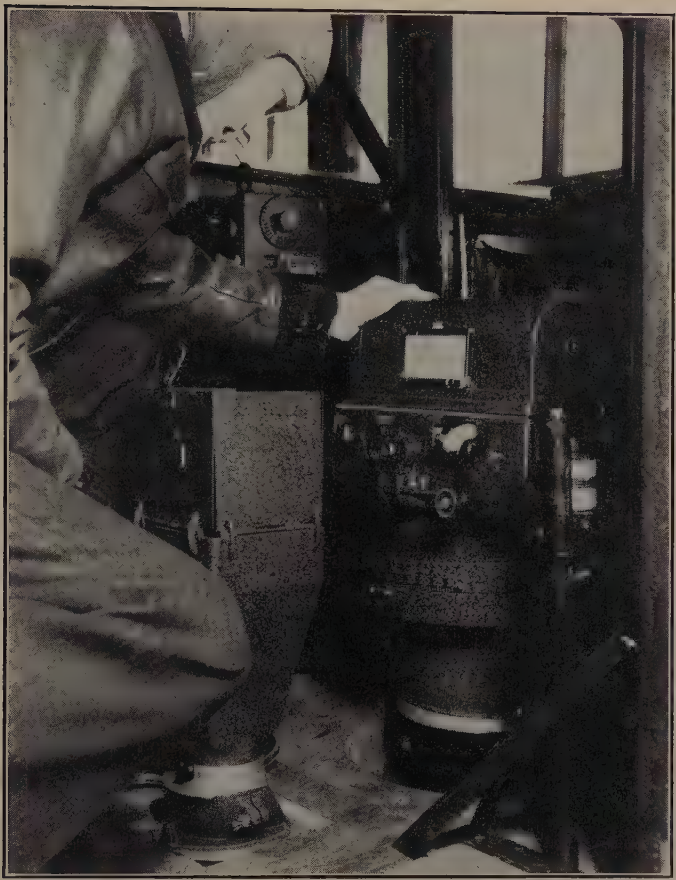

FIG. 1.—TYPICAL INSTALLATION IN AIRPLANE, SHOWING VIEW FINDER AND INTERVALOMETER, FAIRCHILD SINGLE-LENS K-3B AERIAL CAMERA.

FIG. 2.—FAIRCHILD SINGLE-LENS K-3B AERIAL CAMERA AND INTERCHANGEABLE CONES OF FOCAL LENGTHS FROM 6 TO 20 INCHES.

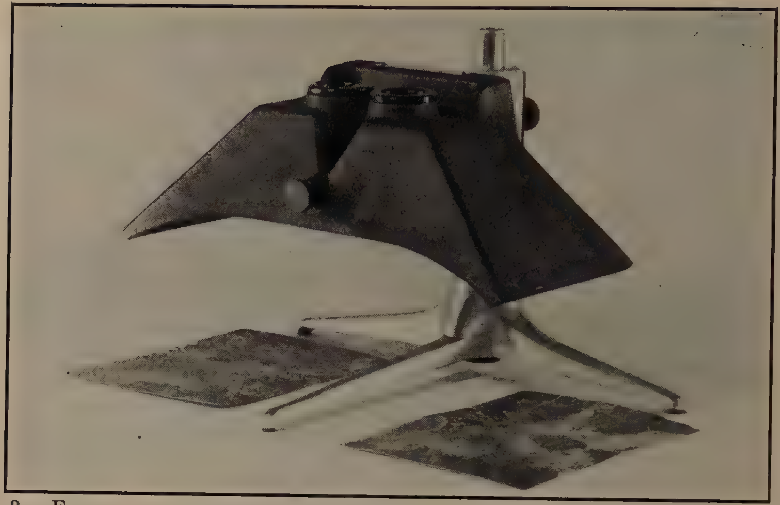

FIG. 3.—FAIRCHILD MAGNIFYING STEREOSCOPE WITH ADJUSTABLE CENTER MIRRORS.

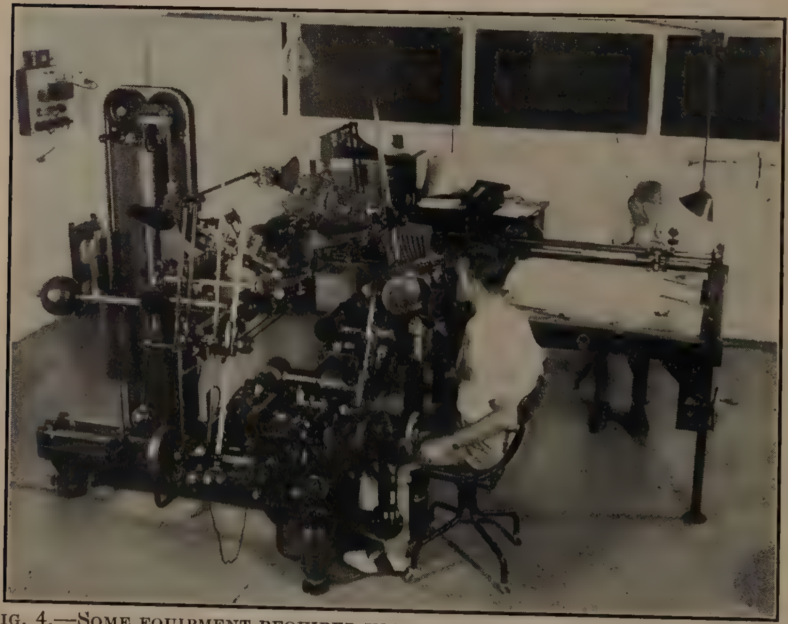

FIG. 4.—SOME EQUIPMENT REQUIRED IN THE MAKING OF AERIAL CONTOUR MAPS.

FIG. 5.—MOTHER LODE AREA NEAR SONORA, CALIFORNIA.
The old stream channel contains auriferous gravels. Lava flowed downstream and covered the gravels. Surrounding area has eroded away and left channels clearly defined. Scale: 1 inch to 1500 feet.

FIG. 6.—YELLOW ASTER MINE, RANDSBURG, CALIFORNIA.
Dikes, faults, outcrops, etc., are visible. Scale: 1 inch to 1000 feet.

FIG. 7.—SAN ANDREAS FAULT, NEAR SAN BERNARDINO, CALIFORNIA.
Some of the parallel faults are water-bearing. Arrowhead Springs, in this vicinity, obtains its water from such a source. Scale: 1 inch to 1400 feet.

FIG. 8.—KELLY MINE, NEAR JOHANNESBURG, CALIFORNIA.
Note vein in which mineralization occurs. Scale: 1 inch to 1500 feet.

Development of Aerial Photographic Equipment

By William H. Meyer, Jr.*

(New York Meeting, February, 1936†)

During the seventeen years Fairchild has been making aerial surveys and aerial photographic equipment many changes and improvements have been made in the equipment and in the technique of using it. Aerial photographic operations have been developed from the crude methods and apparatus of wartime to a science upon which exact engineering calculations can be based. In the early days there was no airplane specially designed to meet aerial photographic requirements. One type of camera was used in all types of operations. Projects were made to conform to the products we could make and deliver. Obviously this entailed definite limitations.

We now have airplanes designed specially for aerial photographic operations in accordance with the conditions encountered in actual service experience. We have cameras for every photographic purpose and our laboratory technique and equipment have been developed to the point where it is possible to make contour maps from vertical aerial photographs with contour intervals as close as five feet.

Therefore, the selection of equipment now can be made in accordance with the time at our disposal, the justifiable cost, the scale desired, and the general requirements of the client. This does not mean, however, that arbitrary deliveries can be set for aerial photographic products. The factors in each project vary and each job must be considered in accordance with the factors involved. This accounts for the fact that products of aerial surveys may readily vary from $1.25 to $200 a square mile.

Sherman M. Fairchild became interested in the design of a radically different aerial camera during the war. At its close he was about ready to demonstrate his design to the War Department. By 1920, he was able to bring out his first completed model. It had a number of radical features, one of which was the first dependable high-speed between-the-lens shutter, which makes for accuracy in aerial photographic work. Another feature was the use of metal throughout, eliminating the wooden parts that had been employed in wartime cameras. Another feature was the interchangeability of parts.

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† See page 559.

One to Nine Lenses

The first models were of the single-lens type, of course. Interchangeable magazines were developed so that fresh supplies of film could be installed in the camera in 15 to 20 seconds. This made it possible to do as much photography as light conditions permit, without losing time to load film in the air or risking fogging the film prior to taking pictures.

Interchangeable lens cones of different focal lengths also were developed. These permit maximum flexibility of the equipment. The focal length of the lens, in inches, divided by the altitude of the plane, in feet, gives the scale of the negatives in inches and feet. For example, to secure a scale of 1 in. to 1000 ft. in the original negatives and use a 10-in. focal-length cone, it is necessary to fly the photographic ship at 10,000 ft. If large-scale photographs are desired, a cone with a longer focal length would be used. This would make it possible for the plane to be flown in the relatively smooth air of higher altitudes and still get the large scale. For instance, at 6000 ft. a 20-in. cone would produce a scale of 1 in. to 300 ft. on the original negative.

A 40-in. cone has been developed for military use, so that large-scale negatives can be secured even from the altitudes that are out of the range of anti-aircraft fire. A 24-in. cone camera is being used by Fairchild Aerial Surveys to secure "close-ups" of various parts of New York City without having to fly so low as to violate Department of Commerce regulations or to involve plane and crew in unnecessary risk.

For mapping purposes cameras are installed vertically on a mount so that the lens shoots through an aperture in the floor of the plane's cabin. For oblique photography the camera is held by the operator, or placed in a flexible mount which is specially designed to absorb the vibration of the plane. When the camera is mounted vertically, a view finder is placed on the floor between the camera and the operator, so that he can see what the camera is photographing and can figure drift and overlap.

Single-lens aerial cameras still are used for aerial surveys. However, it became obvious early that, for certain types of aerial mapping, cameras with more than one lens would be desirable. Such instruments would simplify the problems of control and proper overlap and would embrace larger areas. The first multi-lens camera had three lenses. Later they were increased to four lenses and then to five.

The five-lens camera has been the standard practical multi-lens aerial camera for many years. Improvements have been made in it, however. Since this camera produces a composite print in the shape of a Maltese cross, experiments were made with filling in the blank spaces between the wing prints by making one exposure with the camera and then quickly turning it 45° to take another. The resulting prints fill in a solid octagon. This was not a practical method, so, for the survey of Central New

Mexico in 1935, two five-lens cameras were placed on a single mount, one camera turned 45° from the other. The units were operated simultaneously. This speeded up the survey work immensely, greatly reduced the cost and simplified the control problem.

Carrying out this idea still further, Fairchild has delivered to the U. S. Coast and Geodetic Survey a giant camera with nine lenses mounted in a single unit. It has a longer focal length than has been possible in other multi-lens cameras and embodies a number of revolutionary features. The camera is now in Washington for tests.

Photographic Prints

The wing negatives secured with the multi-lens cameras have to be "rectified" in printing. This is because the wing negatives are secured at an angle and have to be converted to a vertical scale. The process is so exact that each camera has to have its own printer. The printer would not be accurate for another camera.

Aerial pictures are delivered to clients in a number of ways. They can be delivered separately as contact prints, enlargements, or fitted together into mosaics. Sometimes we deliver what we call a "photographic index," in which the overlapping prints merely are stapled together. This type of index map is extremely useful, especially for reconnaissance work, as it forms a rough mosaic of the entire area. Sometimes a line-drawn index map is furnished to indicate the approximate coverage of the original exposures.

One of the most important accessories in the aerial photographic laboratory is the stereoscope. This device makes it possible for the terrain to be viewed in the third dimension. Looking through the stereoscope, it is as though one were looking at a plastic model of the terrain. The stereoscopic principle is the basis of the stereoplanigraph, by which contour maps are drawn from aerial photographs.

Airplane Qualifications

The importance of a suitable photographic airplane became evident in the early days of mapping operations. Existing aircraft fell far short of the ideal attributes and the various attempts to adapt existing airplanes proved only partly successful. One of these adaptations was the installation of an improvised cabin over the rear compartment of an open-cockpit Fokker, to improve in some degree the conditions under which the photographer had to work. The essential attributes of a good photographic airplane are:

1. Good visibility for the pilot so that he may follow the ground features closely directly beneath the airplane, and also be able to see well ahead so as to keep lined up on prominent landmarks.

2. The airplane should be capable of operating at high altitudes, *and we mean really operate*, not just get to altitude and start going backwards. This is not an unusual occurrence, since high winds of 100 miles an hour often are encountered at altitude and if the airplane motor is not of the right type, with sufficient excess horsepower, to provide a flying speed greater than the wind encountered, no photographic work can be accomplished.

3. The airplane should be stable, hold its course, have a wing of high lift characteristic so as to reach altitude in a short time, which is necessary so that the maximum amount of time will be available for actual photographic work.

4. It must have a gasoline capacity for at least five hours of operation.

5. It must be strong in construction so that it will withstand operating from airports near the project, which may be third rate fields or worse.

6. The pilot and photographer should be protected from cold so that they can operate from four to five hours without too much hardship. Also, it is important that they should be able to communicate with each other readily.

To get these characteristics into one ship, Fairchild had to have the plane constructed by his own staff. Besides embodying the features mentioned, the plane became the first cabin plane in this country with the pilot inside the cabin and up front. It also had the advantages of folding wings, which made it easy to store in small spaces and in hangars or on shipboard.

The instrument board included a sensitive altimeter, directional gyro, artificial horizon and other instruments to assist the pilot in flying a straight, level course.

Applications of Aerial Photography

The uses to which aerial photography may be put are almost limitless. Some of them are as follows: tax adjustments, right-of-way negotiations, transmission-line engineering, real estate developments, forest survey, mining geological surveys, industrial engineering, oil prospecting, highway development, harbor surveys, projects for conservation of natural resources, dam and watershed engineering, community improvement planning, and establishment of property lines.

The making of a right-of-way map serves as an example of the method of operation and of the value of aerial surveys. First, the original aerial photographs are secured. A set of contact prints is made for immediate delivery to the client and then a mosaic map is made. Special care is taken to keep the alignment of the various pictures on this map as accurate as possible. On the mosaic map, which may be a strip of considerable length, it is possible for the engineer selecting the route to make

his preliminary survey; for every time he stretches the string across the map between points he is accomplishing as much as he would in a preliminary survey on the ground, either by walking over it or by using transits on long shots. In this way, the cheapest possible route is selected, points of difficulty are readily observed, angles may be put in the line if they are considered important, and the problem can be studied right in the office. All the facts are before the engineers and they do not have to rely upon memory.

Also, it is very important to bear in mind that up to this point everything can be done in a confidential manner. No one need know that the survey has been made; the local inhabitants have not been stirred up or given any warning to increase land prices.

After the preliminary survey has been made by using the map and after the position of the line has been fairly well determined, the next step is to determine the ownership of the various properties that will have to be acquired. This can also be expedited by the use of the pictures, since by studying court records and making local investigations, property data can be plotted more quickly on the pictures than by just making up a sketch map. After this has been done the right-of-way men go into the field with their photographs to acquire the property necessary. The advantages of being able to explain very clearly exactly what land is required, where it is, how far it is from the farm and all important features, are obvious. It has been our experience that when a blueprint or a drawing is submitted to people not familiar with the symbols and representations, they are very reluctant to sell right-of-way based on such maps. However, if they are shown an aerial photograph which they can clearly understand, they can see the trees, the woods, the buildings, the roads, landmarks and features which are familiar to them. They will more readily agree to release a strip of land based on an explanation from a photograph than to grant the same strip of land if all that is presented is a blueprint.

DISCUSSION

[This discussion refers also to the paper that begins on page 560.]

Chairman Marvin.—Listening to Mr. Eliel's paper brought to mind the great progress which has been made in the development of planes, and the engines and equipment going into them in the past 10 years. As Mr. Eliel says, anyone who looked into the possibilities of aviation five or six years ago would be amazed to see what has been accomplished recently.

Another point he emphasized is the degree of specialization which has come into this new business. The data which the Aviation Committee received from its questionnaire survey of companies using aviation pointed out that we *have arrived* at this specialization stage in the use of aviation in mining and petroleum operations. There is no longer any question about the feasibility of using planes for these purposes.

They are being used and different commercial organizations have been developed and are specializing in the different applications.

A. W. Lawson, Chicago, Ill.—Has Mr. Eliel's device been applied to commercial aviation on areas where there is no radio control?

W. H. Meyer.—That particular solar-navigator is specially designed and manufactured for use in a large survey taking place now in New Guinea. It has not been used in this country as yet for any commercial work. It is only within the last two months that the experimentation reached a point where the device was actually used in the air. We expect to use it and hope to use it on securing control for large areas. I don't know whether I quite answer your question.

A. W. Lawson.—Except that I am curious to know whether you expect it to have application for planes flying over the wastes of Canada or other parts of the world where there is no definite radio control such as we have on some of the main established transcontinental airways of this country.

W. H. Meyer.—You mean radial control or radio?

A. W. Lawson.—I may be using the wrong term. What type of device is used for the TWA and American Airways and United in holding their courses—the radio beacons?

W. H. Meyer.—That is radio on which there is an instrument that sends out, say, a series of dots and a series of dashes. If you are right on the beam, it is silent. If you are to the right, you get a series of dashes, and if you are to the left, you get a series of dots in the head phone. That sort of radio control or direction finding is adequate for commercial flying, but it is not close enough for our work. Furthermore, in following a radio beacon, if there is a strong wind, you will arrive at the beacon but you may go in an arc, because you would follow it around, but your plane would be drifting. In our work, if we head into a side wind, we still have to keep the plane going straight on its course to take pictures and turn around and have a parallel strip. What we do hope to do with the solar-navigator is to be able to fly straight lines of pictures between control points; say, astronomical stations. A party or a series group of parties might pick out prominent features and get the astronomical bearing and then by flying straight lines between these control points build a network of control pictures, then with that network a single-lens camera can fly parallel strips back and forth, tying into the previous set of control strips. By securing an astronomical position of these points, maybe 100 miles apart, and having a strip of pictures between the points, forming closed triangles, you can lay down a mosaic, using the detail of the photographs as additional control for every strip of pictures made.

A. W. Lawson.—Then to carry it one step further, and that is the point I had in mind, would it be practicable for commercial aviation if a plane started from some railroad station, say in Canada, and wanted to fly 300 or 400 miles up into some mining section where a prospect was being run? Could the pilot set the solar-navigator and be pretty certain of arriving at the point, irrespective of weather and wind conditions?

W. H. Meyer.—If he had the geographical location of latitude and longitude and the point he wanted to get to, yes.

A. W. Lawson.—There is no limit of these points apart provided the astronomical location of them is known?

W. H. Meyer.—That is right, but there must be sunlight, as the instrument works only in daylight.

H. D. Hibbard, Plainfield, N.J.—How many exposures were taken for each of those pictures, please?

W. H. Meyer.—On some scales, there would be about two pictures to a square mile; that is, there would be about 100 photographs making up the mosaic.

H. D. Hibbard.—How do you make them fit so accurately?

W. H. Meyer.—That is just technique.

Chairman Marvin.—Mr. Meyer, last fall I had an experience in taking some colored aerial movies over the desert in Wyoming. In showing these pictures, I noticed that one could more readily pick out various types of geological formations because of their different colors. Has the question of color been injected into your work as yet?

W. H. Meyer.—No, it has not, because it is difficult to reproduce the color. If you took colored photography, you would have to leave them on lantern slides or on film, because there is no way of getting that color into printed reproduction unless you go to the expense of having a series of color printed plates by means of lithography or electrotypes, and that runs into a young fortune.

Chairman Marvin.—But it could be shown as a film on a slide?

W. H. Meyer.—It could be shown as a film and it could be put on glass. The Department of Agriculture in some of its work, such as gypsy moth control and so forth, has used colored movies. It has never been extensively applied so far by us because a colored mosaic cannot be made, and it is difficult to reproduce them in color.

H. N. Marsh, Los Angeles, Calif.—It is my impression that a few years ago the greatest difficulty was in making prints match up where there was a great difference in elevation of land. Taking an extreme example, if you were flying 10,000 ft. and you came to a ridge 20,000 ft. high, your scale would be approximately double. Has there been a successful way of correcting that?

W. H. Meyer.—That can be corrected by making a series of ratio prints, as we term them, and pyramiding in the making of the mosaic work. If it is necessary and if the project justifies the expense, we can make a mosaic by building up and pyramiding the prints one over the other, and as we get to the higher points of elevation, the degree of enlargement would be less than the enlargement for the prints down at sea level.

A new method has been worked out whereby mosaic maps can be more readily made and more quickly made by using radial control, but not going through the laborious process of tracing out those control points by using tracing paper and assembly. The method has been worked out in Los Angeles, whereby much of the work is done mechanically.

Report of A.I.M.E. Aviation Committee for Year 1936–37

By W. E. D. Stokes, Jr.,* Vice-chairman

(New York Meeting, February, 1937)

The application of aviation to mining and petroleum operations, on the basis of economy and attainment, has become a demonstrated fact.

According to Dominion Government records, 30 Canadian companies engaged in air transportation with northland mining camps, carried 13,000 tons of freight in 1935, and in 1936 carried 16,000 tons. In addition, many mining companies used planes to transport prospectors with canoes, outboard motors, gasoline drums and supplies to remote locations, and one company maintained a plane to take radium-bearing ore from its mine on Great Bear Lake nearly 1000 miles south to the railway at Waterways, Alberta.

Heavy pieces of mining machinery weighing 2 tons are being carried by Ford trimotor planes over the Andes, at an altitude of not less than 15,000 feet.

At Bulolo, New Guinea, 17,500 tons of disassembled gold-dredging equipment has been flown over inaccessible terrain, while supplies and building materials for a community of 1000 whites have been transported by airplane. A 3½-ton shaft was carried on one notable trip, while at another time an automobile, 2 tons of rice and a large safe went over the mountains in one plane. Fig. 1 shows an electric generator that was carried by plane to Bulolo.

Many large American oil companies operated fleets of planes during 1936, ranging from fast, luxurious air liners for executives to elaborate geological prospecting units, ambulance planes, freight carriers and pipe-line repair crews.

According to the U. S. Geological Survey, commercial firms photographed 16,291 square miles during 1935, while the Government compiled from aerial photographs by the Geological Survey in the United States 85,544 square miles. It was found that information could be determined in a fraction of the time required by surface methods.

However, sounding a word of caution, a Commonwealth of Australia government report on aerial survey operations had this to say:

> It is necessary to repeat *ad nauseam* that aerial photographic survey is in no sense a substitute for geological investigation, though it is a very valuable adjunct thereto, and greatly facilitates and accelerates geological mapping. There are many details

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* President, Kessto Corporation, New York, N. Y.

which cannot possibly be discovered from the air. Conversely, however, the ability to comprehend at a glance the whole of the features of a wide area makes it possible

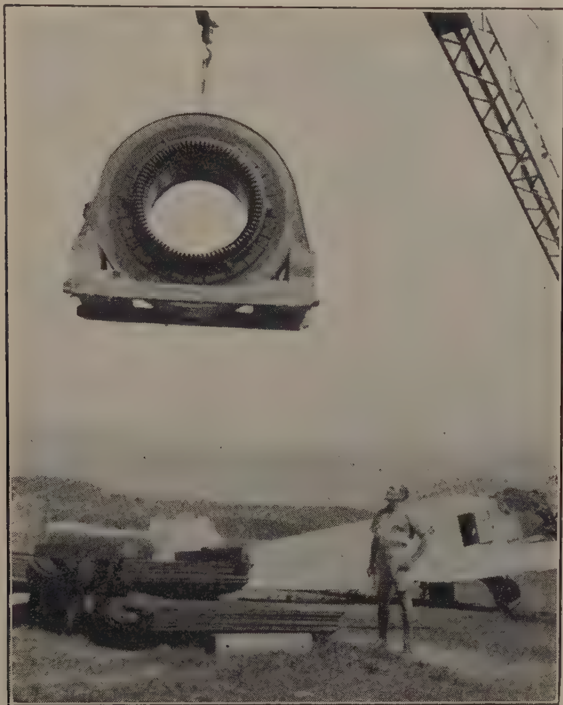

FIG. 1.—ELECTRIC GENERATOR FIELD, WEIGHT 5600 LB., BULOLO, NEW GUINEA.

to detect, from the air, both visually and photographically, many continuities of structure which are entirely lost on the ground.

The Aviation Committee of the A.I.M.E.* during 1936 sent a questionnaire to every known mining and petroleum company using aviation, and corresponded with foreign governments touching their similar activities.

It appears that the various dominions and colonies of the British Empire have nearly all embarked upon ambitious aerial survey programs, using military planes and flyers for this activity, which is in the nature of

* AVIATION COMMITTEE, 1936–1937.

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a mineral census. The success of these programs may be judged from the fact that they are continuing, although the results have been closely guarded, and the expense of little moment. Only Russia and the United States appear to have followed suit. While the Committee has ample evidence at hand that mining and petroleum companies situated in every corner of the world use planes for many purposes, the overwhelming volume of such activity is found to be Canadian and American. Foreign made planes with foreign motors are in the negligible minority. In many instances, such as Bulolo, foreign made planes use American motors, and on an overwhelming number all equipment is from the United States. Consequently, this report might properly be restricted to type applications covering American equipment used in this country and the Dominions, as being a cross section of the whole picture.

Normal Cycle of Use

1. Experience has shown that small planes, equipped with floats and flown by prospectors and geologists, can successfully penetrate virgin territory for weeks at a time, producing results of great value. To mention a few areas—Great Bear Lake, God's Lake, Red Lake, Pickle Crow, Sturgeon Lake and Central Manitoba, in Canada—have been prospected almost exclusively by this method. In such cases, because of a desire for secrecy and flexibility of operation, company or syndicate-owned airplanes are the rule.

Fig. 2.—Bellanca aircruiser seaplane.
Wright geared cyclone engine, 750 hp. Pay load, 3400 lb. Cruising speed, 150 miles per hour.

2. After the ground has been staked by the prospecting plane, the services of a freight plane are required to haul machinery and supplies and personnel to and from the prospect. Planes are used at the same time for locating power sites, conveying tractor trains, carrying mail, doctors,

and investors to the area. Fig. 2 shows the freight plane used for radium-silver ore on the Great Bear Lake waterways, Alberta.

3. If the population of a mining camp and the resulting activity warrants, aviation companies spring up to bid for the business against the company-owned plane. From answers to questionnaires, all the way from New Guinea to Quebec, it appears that this transition invariably occurs, and that the company-owned ship is displaced.

4. Next, the mine company may engage the services of an aerial photographing outfit to make an aerial map or geological survey of its holdings. One large mining concern operated several photographically equipped planes at an expense of nearly $1,000,000 to make a geological survey of an immense area in western Australia.

5. In many areas flying services execute contracts to serve individual mines on a retainer basis, while at others the mines and prospects are served as way stations on a scheduled run.

6. Continuous and successful mining and petroleum operations usually result in the construction of government highways and railroads, so that aviation may be crowded out of the picture. When reserves begin to run low, the operating companies once more take to the air; their prospecting planes race to the scene when reports of a "new area" trickle back to civilization, lawyers and executives secure leases, and geologists stake claims. As frontiers of civilization continue to be pushed back, larger and more self-contained prospecting planes make their appearance. The 1936 Archbold expedition to penetrate the New Guinea wilderness made numerous nonstop flights of 350 miles inland and return, and secured aerial mosaics indicating petroleum salt domes, using a specially equipped $75,000 Fairchild amphibian. The same expedition has purchased a $130,000 Consolidated (Navy type) patrol plane, with a cruising radius of 4500 miles, for its 1937 attempt.

A special twin-engined survey plane capable of photographing nonstop a 50-mile strip measuring nearly 900 miles is now under construction for Hemming and Partners Ltd., mining engineers, by British Aircraft Manufacturing Co., and will operate extensively in South Africa during 1937. The airplane chosen is a B. A. "Double Eagle" of 5000 lb. weight, powered with two Gipsy Six II (205 hp.) engines and American variable-pitch propellors. In an attempt to eliminate ground controls, an automatic pilot to control rudder elevators and ailerons will provide the utmost accuracy in course-keeping, in maintaining a steady height, and in preventing tilt by keeping the airplane on a perfectly even keel while successive exposures are made.

7. Oil companies use planes, not more successfully, but more continuously than mining companies. Usually they have landing fields near by and are not subjected to the inconveniences of thawing (break up) and freezing lakes. These continuous activities include:

DAILY AIRPLANE FLIGHT REPORT
FORM B 1782 FRONT

SOCONY-VACUUM OIL COMPANY
INCORPORATED

AIRPLANE TYPE ______ N. C No. ______ DATE ______ 19__

MECHANICAL DEFECTS NOTED IN FLIGHT	PILOT AND PASSENGERS	TERMINALS	TOTAL FLYING TIME	NO. OF LANDINGS
		FROM ______	HRS. MINS.	
		TO ______		

MAINTENANCE INSPECTION REPORT
FORM B 1782 BACK

DATE ______ 19__

RECORD OF LAST DAILY INSPECTION

ENGINE TIME SINCE OVERHAUL		OIL TIME SINCE CHANGED		ENGINE		LANDING GEAR		MISCELLANEOUS		SIGNATURE OF MECHANIC MAKING INSPECTION
BR'T F'W'D		BR'T F'W'D		ENGINE CONTROLS	☐	WHEELS AND TIRES	☐	TAIL WHEEL ASSEMBLY	☐	
				FUEL LINES AND CONNECTIONS	☐	SHOCK ABSORBERS	☐	EMPENNAGE CABLES & GUIDES	☐	
TIME THIS SHEET		TIME THIS SHEET		DRAIN FUEL STRAINERS	☐	STRUTS AND BRACES	☐	WING STRUTS	☐	
				OIL LINES AND CONNECTIONS	☐	ATTACHMENT FITTINGS	☐			
TOTAL		TOTAL		ELECTRICAL CONNECTIONS	☐		☐			

PILOT'S REMARKS

CORRECTIONS MADE

FUEL (GALLONS)	OIL (QUARTS)	PURCHASED FROM	MAINTENANCE WORK (SHOW REPAIRS MADE, PARTS REPLACED AND SUCH REMARKS AS WILL EXPLAIN THE CONDITION OF THE AIRPLANE)
TOTAL			

a. Transportation of executives from office to rig.
b. Transportation of emergency equipment and personnel.
c. Policing pipe lines.
d. Sales and advertising.
e. Surveying.
f. Geologizing.
g. Ambulance service.

Useful Load

Modern single-engined amphibians will carry a useful load of about 33 per cent of their gross weight, twin-motored amphibians 32 per cent, single-engined land planes from 37 to 43 per cent, multimotored land planes 34 to 38 per cent. Useful load is defined as the weight of the minimum crew necessary to the operation of the airplane, commonly consisting of the pilot only, fuel, oil and pay load.

Fig. 4.—Curtiss-Wright Condor cargo plane.

It is interesting to compare the ratios of useful load carrier per horsepower of the various airplanes, with the thought that the horsepower is a handy indication of the flying cost.

Modern single-engined amphibians carry between 3 and 4 lb. of useful load per engine horsepower, multiengined amphibians only slightly less; single-engined land planes carry between 4 and 5 lb., and multimotored land planes about 4 lb. of useful load per horsepower. Using these figures, an estimate of the size of airplane required can be made in any type of ship necessitated by the particular operating problem.

Operating Costs

While improvements have been made in speed, landing and take-off characteristics, generally at the expense of rugged simplicity of design,

a

b

FIG. 5.—CURTISS-WRIGHT CONDOR CARGO PLANE.
a, loading; *b*, interior view.

from a comparative operating cost basis it can be said that the older and slower types of planes are still relatively desirable for mining operations. The increased performance of newer types could probably produce a 10 per cent saving in fuel consumption and this item in turn would account for about 20 per cent of the total operating cost.

A study of all the types and sizes of planes in use would apparently lead one to the same conclusion, viz.:

1. The total yearly cost of operating a plane in this type of work on an average full-time basis approximately equals the initial cost of the plane.

2. Two-thirds of the total cost consists of depreciation (25 per cent), pilot's and mechanic's salaries, insurance, and field maintenance.

3. One-third of the total cost is accounted for by gasoline, oil, repairs and parts.

Companies paying liberal salaries for experienced pilots and liberal allowances for motor-part replacements apparently turn in economical performance records.

The opinion has gained prevalence that airplanes in mining and petroleum operations are operated in a slam bang manner that reflects a psychological approach to flying by men accustomed to life in the wilderness, constant use of heavy explosives, gushing oil wells, grinding mills, etc. While high depreciation allowances and maintenance figures are noted, comparatively few fatalities follow. Planes operated by the companies that have daily flight and maintenance-inspection reports show commendable cost figures. Form B1782 in use by Socony-Vacuum Oil Company is a valuable example (Fig. 3).

B. Allison Gillies, Consulting Aircraft Engineer, Naval Aircraft Factory, Hicksville, N. Y., has prepared the following seven estimates of

TABLE 1.—*Operating Costs Stinson Reliant Lycoming* 245-*hp. Motor*, 1937

(Assume plane is to be flown 420 hr. per year or 35 hr. per month.)

Fixed Operating Costs:		
Gasoline: 25¢ per gal. × 16 gal. per hr.	$4.00	
Oil: 35¢ per qt. × 1½ qt. per hr.	0.52	
	$4.52 per hour	
Costs per Month:		
Fixed operating costs: $4.52 per hr. × 35 hr.		$158.20
Hangar rent		35.00
Pilot-mechanic salary		250.00
Repairs, maintenance and reserve for motor overhauls		52.50
Insurance (includes fire, public liability, property damage, windstorm and land damage, but does not include passenger liability or crash)		45.00
Total operating costs per month		$540.70

Depreciation: $2,500.00 per year or $208.33 per month

Cost of Stinson Reliant with Lycoming 245-hp. motor complete with floats and corrosion proofing, but without blind-flying equipment or radio equipment, approximately $12,750.00.

operating costs for current models (Tables 1 to 7). Pilots' and mechanics' salaries are subject to wide variations, depending upon the section of the country in which a ship is based. Figures used in these tables are normal for metropolitan New York area.

TABLE 2.—*Operating Costs Stinson Reliant Wright, 320-hp. Motor Hamilton Controllable Propeller,* 1937

Operating costs *as low as*: 9.23¢ per airplane mile (5 people)
1.85¢ per passenger mile

	Per Airplane Mile (5 People)	Per Passenger Mile
First cost (F. A. F. Wayne, Michigan)		$12,785.00
Cruising speed[a] (225 hp.) at optimum altitude, miles per hour		161
Direct Operating Costs:		
Fuel: 18 gal. (U. S.) per hr. at 25¢ per gal	$0.0279	$0.00558
Oil: 2 qt. per hr. at $1.40 per gal	0.0044	0.00088
Maintenance and overhaul[b]	0.0152	0.00304
Total direct cost	$0.0475	$0.0095
Fixed Charges:		
Hangar: at $25 per month	$0.0038	$0.00076
Depreciation (see below)	0.0339	0.00678
Insurance (see below)	0.0071	0.00142
Total fixed charges	$0.0448	$0.00896
Combined total operating cost[a]	$0.0923	$0.0185

[a] Based on 500 hr. of operation per year. The direct unit operating costs are independent of the hours of operation per year.

The unit fixed charges vary with the amount of flying per year. Therefore, if the operator flies less than 500 hr. per year, the combined total operating cost would be 10.35¢ per airplane mile for 400 hr. per year, and 12.23¢ per airplane mile for 300 hours.

[b] The cost of maintenance and overhaul is based on the average of a large number of actual operating cost figures and is believed to represent the case of the average private operator. The cost figures given above include charges for inspection, as well as top and complete overhauls of the engine at periods recommended by the manufacturer.

TABLE 3.—*Operating Costs Waco Custom Cabin Jacobs 285-hp. Motor,* 1937

(Assume plane is to be flown 420 hr. per year or 35 hr. per month.)

Fixed Operating Costs:		
Gasoline: 25¢ per gal. × 17 gal. per hr	$4.25	
Oil: 35¢ per qt. × 1½ qt. per hr	0.52	
	$4.77 per hour	
Costs per Month:		
Fixed operating costs: $4.77 per hr. × 35 hr		$166.95
Hangar rent		35.00
Pilot-mechanic salary		250.00
Repairs, maintenance and reserve for motor overhauls		52.50
Insurance (include fire, public liability, property damage, windstorm and land damage, but does not include passenger liability or crash)		45.00
Total operating costs per month		$549.45

Depreciation: $2,500.00 per year or $208.33 per month

Cost of Waco Custom Cabin with Jacobs 285-hp. motor complete with floats and corrosion proofing, but without blind-flying equipment or radio equipment: $11,000.00 approximately.

TABLE 4.—*Operating Costs Waco Standard Cabin 225-hp. Jacobs Motor, 1937*

(Assume plane is to be flown 420 hr. per year or 35 hr. per month.)

Fixed Operating Costs:		
Gasoline: 25¢ per gal. × 14 gal. per hr.	$3.50	
Oil: 35¢ per qt. × 1 qt. per hr.	0.35	
	$3.85 per hour	
Costs per Month:		
Fixed operating costs: $3.85 per hr. × 35 hr.		$134.75
Hangar rent		35.00
Pilot-mechanic salary		250.00
Repairs, maintenance and reserve for motor overhauls		43.75
Insurance (includes fire, public liability, property damage, windstorm and land damage, but does not include passenger liability or crash)		40.00
Total operating costs per month		$503.50

Depreciation: $2000.00 per year or $166.66 per month

Cost of Waco Standard Cabin with Jacobs 225-hp. motor complete with floats and corrosion proofing, but without blind-flying equipment or radio equipment: approximately $7,500.00.

TABLE 5.—*Operating Costs Fairchild 24, Seaplane, 1937*

(Assume plane is to be flown 420 hr. per year or 35 hr. per month.)

Fixed Operating Costs:		
Gasoline: 25¢ per gal. × 8 gal. per hr.	$2.00	
Oil: 0.35¢ per qt. × 1 qt. per hr.	0.35	
	$2.35 per hour	
Costs per Month:		
Fixed operating costs: $2.35 per hr. × 35 hr.		$ 82.25
Hangar rent		30.00
Pilot-mechanic salary		250.00
Repairs, maintenance and reserve for motor overhauls		35.00
Insurance (includes fire, public liability, property damage, windstorm and land damage, but does not include passenger liability or crash)		40.00
Total operating costs per month		$437.25

Depreciation: $2,000.00 per year or $166.66 per month

Cost of Fairchild on floats with complete corrosion proofing, but without blind-flying equipment or radio: approximately $7,500.00.

TABLE 6.—*Operating Costs Sea Bird Amphibian,* 1937

(Assume plane is to be flown 50 hr. per month, or 6650 miles.)

OPERATING COSTS

	Per Month	Per Mile Cents
Those varying with distance flown:		
Gasoline: 25¢ per gal. for 850 gal.	$212.50	3.20
Oil at $1.40 per gal. for 12½ gal.	17.50	0.26
Repairs and maintenance	45.00	0.68
Insurance (public liability at 85¢ per flying hour, $5 per $10,000)	42.50	0.64
Total costs per month that vary with distance flown for 50 hr.	$317.50	4.78
Those which remain approximately constant:		
Storage (hangar rent)	$ 35.00	0.53
Depreciation (8 yr. life with $18,500 original cost)	194.00	2.92
Insurance (ordinary fire insurance, not including fire resulting from accident, 3 per cent of value per annum)	46.25	0.69
Total costs per month that remain approximately constant	$275.25	4.14
Total monthly operating cost including depreciation for 50 hr. flying	$592.75	8.92
If a pilot is regularly employed at	$300.00	4.51
Total cost including pilot and depreciation for 50 hr. flying per month	$892.75	13.43

Cost of airplane, $18,500.00

TABLE 7.—*Grumman G-21 Amphibian,* 1937

(Two 420-hp. Pratt & Whitney engines. Cost of operation based on 500 hr. flying per year.)

Gasoline: 42 gal. per hr. at 30¢ per gal	$6,800.00
Oil: 2 qt. per hr. at 35¢ per qt.	350.00
Reserve for major motor overhauls, $2.00 per hr.	1,000.00
Motor maintenance, $1.00 per hr.	500.00
Airplane maintenance, $1.00 per hr.	500.00
Hangar rent, $60.00 per month	720.00
Insurance (includes fire, windstorm, theft, land damage, property damage with $5,000 limit, and public liability with $10 per $20,000 limit)	1,600.00
Pilot's salary, $100.00 per week	5,200.00
Mechanic's salary, $40.00 per week	2,080.00
Compensation insurance, etc. (estimated)	1,200.00
	$19,950.00
Depreciation on 5-year basis	$10,000.00
Total cost per year	$29,950.00
Cost per hour	$ 59.90

Tables 8 and 9 were prepared by Hugh M. Wolflin and submitted by courtesy of the U. S. Bureau of Mines. Table 10 was supplied by James Readdig, engineer at the Seversky aircraft factory.

TABLE 8.—*Flying Costs for Six Lockheed Vegas, Operated by Petroleum Companies during* 1931 *and* 1932[a]

	1	2	3	4	5	6	Approximate Average
General:							
1. Purchase price (estimated)	$19,000	$13,000	$16,589	$17,651	$21,202	$20,706	$18,025
2. Engine: Wasp, hp	420	420	425	425	425	?	
3. Flying hours prior to period	1,295		92	269(?)	312(?)		
4. Year in which operated	1932	1931	1931	1931	1931	1931	
5. Flying hours during period	309	281	659	178	272	531	371
6. Cruising speed, given or assumed, miles per hr	140	160	140	140	140	140	143
7. Mileage, actual or estimated	43,260	44,960	92,260	24,920	38,080	74,340	52,970
Indirect or fixed costs:							
8. Depreciation at 25 per cent per year	$ 4,750	$ 3,250	$ 4,147	$ 4,413	$ 5,300	$ 5,177	$ 4,506
9. Insurance, license, etc	4,224		4,005	475	475	4,572	
10. Salaries of pilot and mechanic	6,401	528		0	0	6,460	
11. Storage, including field rental	872	660	571			1,059	
12. Hangar-field maintenance, amortization		668		3,154	3,154		
13. Miscellaneous indirect costs	960		289	580	697		
14. Total indirect costs	$17,207	$ 5,106	$ 9,012	$ 8,622	$ 9,626	$17,268	$11,140
15. Indirect cost per flying hour	55.686	18.172	13.675	48.438	35.390	32.520	30.027
16. Indirect cost per airplane-mile	0.398	0.114	0.098	0.345	0.253	0.232	0.210
Direct or variable costs:							
17. Gasoline	$ 1,923		$ 2,144	$ 741	$ 1,247	$ 2,766	
18. Oil and grease	296		(see 17)	(see 17)	(see 17)	725	
19. Expenses of pilot and mechanic						1,526	
20. Airplane repairs						(see 23)	
21. Engine repairs and overhauls				75	591	(see 23)	
22. Materials and repairs (no details)	201			2,185	943	(see 23)	
23. Repairs, outside	498		4,956			7,588	
24. Miscellaneous, including outside storage				99	221		
25. Total direct costs	$ 2,918	$ 4,804	$ 7,100	$ 3,100	$ 3,002	$12,605	$ 5,588
26. Direct cost per flying hour	9.444	17.096	10.774	17.416	11.037	23.740	15.062
27. Direct cost per airplane-mile	0.068	0.107	0.077	0.124	0.079	0.170	0.105

[a] Prepared by Hugh M. Wolflin, and submitted by courtesy of the United States Bureau of Mines. Some of the cost figures, contributed for periods of less than 12 months, were converted to a 12-month period.

TABLE 9.—*Flying Costs for Cabin Planes of Four, Five and Six Places, Used for Petroleum and Similar Work*[a]

	1	2	3	4	5
	Fairchild F-71 5-place	Bellanca Skyrocket 6-place	Travel-Air S-6000-B 4-place	Stinson 4-place	Stinson 4-place
General:					
1. Purchase price (estimated)	$19,340	$17,950	$14,215	$4,995	$4.995
2. Engine name, hp	Wasp, 425	Wasp, 420	Wright, J-6-9	Lycoming, 215	Lycoming, 215
3. Year in which operated	1931	1931–'32	1931	1931–'32	1931–'32
4. Flying hours during period	469	450	604	435	319
5. Cruising speed, given or assumed, miles per hr	100	130	100	103	103
6. Mileage, actual or estimated	46,900	58,500	60,400	44,805	32,857
Indirect or fixed costs:					
7. Depreciation at 25 per cent per year	$ 4,835	$ 4,488	$ 3,554	$1,249	$1,249
8. Insurance, license, etc	265	988			
9. Salaries of pilot and mechanic	(none)	2,708	2,103	870	
10. Storage, incl. field rental	(see 12)	585	914	360	240
11. Hangar-field maintenance, amortization	3,847		394		
12. Miscellaneous indirect costs	1,623				
13. Total indirect cost	$10,570	$ 8,769	$ 6,965	$2,479	$1,489
14. Indirect costs per flying hour	22.538	19.487	11.531	5.699	4.668
15. Indirect costs per airplane-mile	0.225	0.150	0.115	0.055	0.045
Direct or variable costs:					
16. Gasoline	$ 2,734	$ 1,821		$1,044	$ 848
17. Oil and grease	(see 16)	(see 16)		109	171
18. Materials and repairs	2,523	1,130		451	1,163
19. Miscellaneous, including outside storage	689	110			129
20. Total direct cost	$ 5,946	$ 3,061	$ 6,136	$1,604	$2,311
21. Direct costs per flying hour	12.678	6.802	10.157	3.687	7.245
22. Direct costs per airplane-mile	0.127	0.052	0.102	0.036	0.070

[a] Prepared by Hugh M. Wolflin, and submitted by courtesy of the United States Bureau of Mines. Some of the records on which this tabulation was based were incomplete. In a few instances astimates were inserted where actual costs were not available.

TABLE 10.—*Estimated Operating Costs of Bellanca Cargo Aircruiser*, 1937

Cost of aircruiser (including controllable pitch prop)	$37,660.00
Cost of engine	10,400.00
Net cost of airplane only	$27,260.00
	PER HOUR
Operating cost at 1500 hr. per year:	
Depreciation, airplane only, three-year basis—$9,086.66 per year	$ 6.06
Depreciation, engine only (1500-hr. write-off)	6.93
Insurance, 15 per cent per year on $37,660.00	3.76
Gasoline, 43 gal. per hour @18¢	7.73
Oil, 6 quarts @15¢	.90
Pilots (2) at $4,000 per year	5.33
Maintenance: motor $2.00	
plane $2.00	4.00
Total cost per hour	$34.71
Cost per mile at 160 miles per hour cruising speed, cents	21.7
Cost per ton-mile, cents	10.85

SUMMARIES OF VARIOUS TYPICAL OPERATIONS

1. *Compania Minera Agua Fria, S. A., Honduras, Central America*, 1934. Hauling machinery to mine, Ford Tri-motor, cost $22,000, including strengthening for maximum loads. Three 420-hp. Wasp B engines. Costs per plane flying hour, $70.00.

The airplane, in moving heavy machinery into a remote district, was the best solution to the transportation problem.

2. *Continental Oil Company, Ponca City, Okla.*, 1935. Transportation of personnel. Three planes cost $87,874.73. Lockheed Electra 10-A, two Wasp engines, cost $110.00 per hour. Lockheed Vega, two Wasp engines, cost $62.00 per hour. Beechcraft biplane, one Wright Whirlwind, cost $68.19 per hour.

3. *Seismograph Service Corporation, Tulsa, Okla.*, 1936. Consulting geophysical prospecting. Stinson Model S. R., Lycoming, 225-hp. Cost $6,000 and $12.00 per hour. Waco S-6, Jacobs 225 hp., cost $5,400 and $12.00 per hour.

4. *Humble Oil and Refining Co., Houston, Texas*, 1934. Sales work. Waco C. Cost $9,000 and $15.00 per hour.

5. *Kansas Power and Light Co., Salina, Kansas*, 1935. Transportation of personnel, aerial mapping and pipe-line surveys. Stinson model S.R. 5, Lycoming 225-hp. Cost $6,400 and $13.00 per hour.

6. *MacMillan Petroleum Corporation, Los Angeles, Calif.*, 1934. Transportation of personnel. Bellanca Skyrocket Wasp costs $14.00 per hour, including pilot's pay but not insurance.

7. *Oil Well Drilling Co., Dallas, Texas*, 1934. Drilling contractors. Cavaliers' Lambert cost $3,185.00 and $4.81 per hour.

8. *Phillips Petroleum Co., Bartlesville, Okla.*, 1936. Transportation of personnel, aerial mapping and geologizing, pipe-line, water-supply and road surveys. One Lockheed, Cyclone, Orion; one Lockheed, Wasp, Vega; one travel Air Wasp 6-place; one travel Air Wasp 4-place; two Cavaliers, Lambert, 2-place; one Boeing 247-D, executive, powered with two gear Wasp engines. Cost ranges from $4,000 to $76,000 each; cost per mile flown from 7¢ to 60¢.

9. *The Pure Oil Company, Columbus, Ohio*, 1936. Advertising, sales promotion and transportation personnel. Waco C, Continental 225-hp., Bellanca Pacemaker, Wasp Jr., 300-hp., Fokker Standard, Wright 330-hp. Cost $6,000 to $17,000.

Waco cost per hour, $13.96; Fokker cost per hour, $15.44; Bellanca cost per hour, $37.72.

10. *Socony-Vacuum Oil Co., Inc., New York, N. Y.*, 1936. Transportation of personnel and aerial mapping. One Stinson Reliant, Lycoming; one Beechcraft, Wright; one Bellanca, Pratt & Whitney; one Travel-Air, Jacobs; one Monospar Jubilee, Popjoy. Cost from $8,000 to $27,000. Average cost per flying hour stated to be $8.25 including depreciation, insurance, gas, oil, maintenance and storage. Planes are renewed every three years.

Fig. 6.—Grumman G-21 amphibian.

11. *Standard Oil Company (New Jersey) Quirequire, Venezuela*, 1936. Transportation of material and personnel, prospecting, mapping, geologizing, pipe-line, water-supply and road surveys. One Sikorsky, S-38 Wasp; one Fairchild, F-71 Wasp; one Junkers Hornet; one Douglas Dolphin, Wasp; one Stinson Lycoming. Cost $60,000 for one on down to $25,000. Fairchild: cost per hour $33.00 for plane and $25.00 for crew. Sikorsky: cost per hour $87.00 for plane and $32.00 for crew. Practice is to lease facilities from commercial companies, except for Venezuela operation.

12. *Standard Oil Company (Ohio), Los Angeles, Calif.*, 1936. Sales, personnel, mapping and pipe-line surveys. Waco de luxe cabin, Wright 285-hp. Cost $12,500.

Cost per hour was $21.25, including all costs, directly or indirectly, in course of operating plane. Insurance cost per hour was $4.37.

13. *Union Oil Company of California, Los Angeles, Calif.*, 1936. Sales, geologizing, mapping and personnel. Stinson Reliant, Lycoming 300-hp. Cost $10,000 and $15.00 per hour, pilot's salary and expenses not included.

14. *United Gas Public Service Co., Houston, Texas,* 1936. Transportation of personnel and aerial photographic work. Vultee V-1A. Cost $42,000 and $40.00 per hour.

15. *American Potash & Chemical Co., Trona, Calif.*, 1935. Geological survey, under direction of Fairchild Aerial Surveys, Inc. Fairchild 71, Pratt & Whitney. Cost $18,000 new, and $4,000 to $6,000 used. Equipped for photography. Overhead expense of maintaining and holding equipment ready for limited amount of suitable photographic weather is major expense.

16. *Anglo American Corporation of South Africa, Ltd., Johannesburg, Union of South Africa*, 1935. Transportation of personnel, prospecting, aerial mapping and geologizing. Officials of corporation utilize either Imperial or Union Airways, and hire from African Air Transport, Ltd. Corporation also uses Aircraft Operating Co., Ltd., for photographic work.

17. *Bulolo Gold Dredging, Ltd., New Guinea,* 1935. Aviation is used for all purposes of transportation, prospecting and mapping. Three Junkers, all metal G-31 trimotor 600-hp. Hornet, Pratt & Whitney engines. Total cost $325,794. Planes operated by Guinea Airways, Ltd., under a management contract.

Air line from coast airdrome at Lae to Bulolo is about 35 miles. Bulolo airdrome is 2500 ft. above sea level, and a mountain range 5000 ft. high rises between Lae and Bulolo. The cost per ton transported, including drome maintenance, general charges, flying costs, cargo handling from shipside to airdrome, airplane insurance, and management fee to Guinea Airways, but *excluding amortization,* is about $60.00 a ton; this figures at about $1.71 per ton-mile. The cost per ton-mile is comparatively high, because the distance is so short. The cost per ton transported would be little more than at present if the distance were doubled. There is little, if any, back loading from Bulolo to Lae.

18. *Consolidated Mining & Smelting Co. of Canada, Ltd., Trail, B. C.*, 1935. Transportation of material, personnel, prospecting and geologizing. Two Fairchild 71-C, Wasp; one Curtiss Robin, Challenger 165-hp.; one De Haviland Dragon seaplane; one De Haviland Puss Moth; two De Haviland Gypsy Moths. Total cost new $92,755. Average cost per flying hour $34.84. Planes average two to three years, pilots are commercially trained; three are mining engineers.

19. *Dandazi G. M. and Estate Company, Ltd., Bindura, Southern Rhodesia*, 1934. Aerial mapping and geologizing. Work was carried out under contract, two square miles of claim at a cost of £85. Actually, about five square miles was obtained at an extra cost of £25.

20. *God's Lake Gold Mines, Ltd., Winnipeg, Manitoba*, 1936. Transportation of material, personnel, prospecting and geologizing. Uses Canadian Airways, Ltd., and Wings, Ltd., which operate 10 planes each in God's Lake district.

21. *Howey Gold Mines, Ltd., Red Lake, Ontario*, 1936. Transportation of material, personnel, prospecting and geologizing. Three flying companies are operating 12 planes at Red Lake. Costs generally figured at 40¢ per mile or $40.00 per flying hour.

22. *Hudson Bay Mining and Smelting Co., Ltd., Manitoba*, 1936. Transportation of express and perishables. There are three commercial lines at this depot. Generally speaking, air transportation has only one advantage—speed. It costs far more than other methods.

23. *The London and Rhodesian Mining and Land Co., Ltd., Salisbury, Southern Rhodesia*, 1935. Gold bullion is brought in by company's plane twice a month,

from three of the mines of its group, which are some distance from the railway. All visits of inspection by engineers to company's mines or alluvial workings, visits of inspection by general manager and local secretary to company's ranches and agricultural estates; visits of inspection of group auditor to mines, ranches and agricultural estates; transfer of staff from one mine or estate to another; transfer of technical staff in case of breakdown on mines, etc. Aerial mapping of alluvial areas has been carried out for company by Aircraft Operating Company (Africa) Pty, Ltd. Waco Cabin biplane, Continental R.670, 210-hp. Cost £1,800. Cost per flying hour £6. 0. 8d.

24. *McIntyre Porcupine Mines, Ltd., Schumacher, Ontario*, 1936. Transportation of material, personnel, prospecting and geologizing. Bellanca, combination sea and land plane, Wright Whirlwind 330-hp. Cost $21,978, and $44.13 per hour. Pilot is mining engineer.

25. *Noranda Mines, Ltd., Noranda, Quebec*, 1936. Examination of prospects. Contract subject to tonnage and flying time, varying from 3¢ to 12¢ per pound. Costs $15.00 to $35.00 per hour.

26. *Prospectors Airways Co., Ltd., Toronto, Ontario*, 1936. Mining and prospecting Fairchild 82 Wasp 520-hp. Cost $21,000 and $50.00 per hour. Gipsy Moth 120-hp. Cost $8,000 and $15.00 per hour. Pilot and prospectors are geologists and frequently spot mineral deposits from the air. Noorduyn Norseman, cost $24,000, Wright 420-hp., not yet delivered.

27. *Tokess, Inc., New York, N. Y.*, 1936. Prospecting. Waco F-4 Jacobs 225-hp. Cost $6,000. $5.00 per hour flying cost; $2.00 hangar; $1.00 parts, $1.00 depreciation; $1.00 insurance. Total $10.00, or 7¢ a mile.

28. *Treadwell Yukon Co., Mayo, Yukon Territory*, 1936. Prospecting. Fairchild Model FC-2-W, Wasp 420-hp. Cost $25,000. Bellanca Skyrocket, Wasp 420-hp. Cost $31,000. De Haviland, Cirus. Cost $9,000. Operating cost between $25.00 and $80.00. Pilots are mining engineers commercially trained.

29. *African Air Survey Co., Ltd., Johannesburg, South Africa*, 1936. Geological interpretation using De Haviland, Leopold Moths and Puss Moths, 120-hp. Gipsy Six. Cost per hour £2.10.0.

30. *M. & C. Aviation Co., Ltd., Prince Albert, Saskatchewan.* Prospecting. Two Fairchilds, one Stinson, one Waco. $30.00 cost per flying hour in summer; $25.00 cost per flying hour in winter. Pilots are mining engineers.

Aerial Survey Notes

Since the technique of aerial survey for geological purposes is fairly new, methods of observation and interpretation are largely lacking. As nearly all the established mining and oil operations have now been photographed, the pictures in many instances being readily available through the U. S. Bureau of Mines, the U. S. Geological Survey, the Fairchild Aerial Camera organization, or the American Institute of Mining and Metallurgical Engineers, it is earnestly hoped that type cases may soon be published in textbook form, accompanied in every case by ground plans, so that students can begin to appreciate the significance of details.

A few of the methods used heretofore with success, are included:

1. *Soil Patterns* are useful in locating salt domes. Evidence of incipient parching points to horizontal stratification.

2. *Difference in Timber Growth.*—Distinction in tree growth usually reveals very clearly the main features of geological structure.

3. *Stratification and Faulting,* in arid ground, appear with the sharpness of detail of a textbook diagram.

4. *Difference in Resistance to Weathering* often indicates sediments and volcanic rocks.

5. *Granite Outcrops* are easily recognized and "fingers," when found, sometimes point to main breaks of ore-bearing lodes.

6. *Tree Patterns* in some places take the form of ovals, lemniscates or lozenges, indicating approximate horizontality of stratification, in an area of low relief.

7. *Shadow Effects.*—In an area of moderate relief, the strike of the rocks appears although concealed in underbrush. By using the directions of the shadows of trees and other objects in the print, it is possible to determine the orientation of the picture independently of compass bearings, if the exact moment of exposure and the latitude and longitude are approximately known.

8. *Contour Lines* prepared with the aid of stereopticon camera equipment are the most important modern developments of aerial geology.

9. *Transverse Structures.*—Volcanic dikes are easily traced across main ridges of a mountain complex.

10. *Anticlines and the Synclines* are usually conspicuous.

11. *Change in Physiography,* produced, for instance, by transition from pre-Cambrian schists to the granites is striking, the latter areas being much less rugged in character.

12. *Contact Line.*—Aerial maps made after the failure of the St. Francis Dam, California, clearly show the contact line running through the dam site.

13. *In Situ.*—At a California gold-dredging operation, aerial photographs revealed the area to be a decomposed rock mass carrying a mass of quartz stringers.

14. *Tree Types.*—A certain type of tree or other vegetation has been found to be indigenous to regions with good copper prospects. Aerial maps have revealed this vegetation, thereby directing exploration parties to regions which justify investigation.

15. *Oil Seepages* are revealed by their effect on the foliage. The *axis* of the anticline was clearly revealed by the strike of the beds along the streams.

16. *Faults* were revealed through areas such as orchard land by difference in soil coloration depicted in aerial maps on opposite sides of the fault.

17. *No Rock Outcrops.*—Where profound weathering has caused complete decomposition of the rock formation, leaving deep soils *in situ* to mark the former distribution of rock types, photographed from the air, they stand out with all the sharpness and clarity of a color map.

18. *Measurement of Dips.*—This is capable of being done from the air in country difficult of access, and is valuable in determining closure in oil

prospecting. The method is to fly in circles around the outcrop, rolling down one wing until the surface appears to coincide or parallel the formation. The bank of the plane, as indicated in degrees upon the Sperry gyro horizon, will then be the dip of the outcrop.

19. *Swirls.*—Certain overlapping folded structures have been identified as collecting grounds for petroleum, and are readily noticeable from the air, taking the form of gigantic ellipses.

20. *Hot Springs* may be noted from the air in arid country and are held to indicate the existence of fault lines.

Future Developments

Wave Lengths.—Prof. Paul MacClintoch, of Princeton, has indicated the possibility of identifying rocks from the air by measuring the wave length of their light reflection. He suggests: (1) the use of the spectroscope and (2) the taking of four simultaneous pictures from aerial cameras equipped with A, B and C Wratten color-separation filters. Experiments have been conducted and interesting results obtained.

Radioactive Emanations when measured on a Geiger counter may indicate the presence of igneous intrusions.

Latest Models

The following named makes and planes are submitted, on the basis of their service record; the latest model only being specified.

A. *Passenger and Cargo*

Bellanca Aircruiser cargo land or seaplane, one 750-hp. Hornet, 575-hp. Cyclone
Bellanca Senior Pacemaker freight land and seaplane, one 550-hp. Wasp, one 420-hp. Whirlwind
Bellanca Senior Skyrocket land and seaplane, one 550-hp. Wasp, one 420-hp Whirlwind
Curtiss-Wright Condor cargo
Fairchild Model 91 Amphibian, one 575-hp. Hornet or Cyclone
Fairchild XC-31 cargo transport, one 575-hp. Cyclone
Grumman G-21 commercial amphibian, two 420-hp. Wasp Jr.
Junkers W-34 cargo, three 550-hp. Wasps, three 700-hp. Hornets
Lockheed Altair cargo, one 550-hp. Wasp
Lockheed Electra, two Wasps or Wasp Jr., two Whirlwinds
Lockheed Model 12, Wasps, Whirlwinds or Menascos
Lockheed Orion, one Wasp
Lockheed Vega, one Wasp
Noorduyn Norseman freighter, one 420-hp. Wasp Jr.
Northrop Delta mail plane, one 525-hp. Hornet
Northrop Delta transport, one 575-hp. Cyclone
Northrop Gamma mail plane, one 575-hp. Cyclone
Stinson Model B, two Lycomings
Vultee V-1A, one 735-hp. Cyclone

B. *Personnel and Material*

Bellanca De Luxe Skyrocket
Bellanca Pacemaker Executive, one Wasp or Whirlwind
Fairchild 24 Model C8-C, one 145-hp. Ranger
Fairchild Highlift Tail No. 71, one Wasp
Fairchild 45, one Whirlwind
Stinson Reliant, one Lycoming
Waco Model CJC-S, one Wright Whirlwind
Waco Cabin, one Jacobs, one Continental
Waco F, one Jacobs, one Continental, one Whirlwind

C. *Special*

Grumman J-F2 Navy photographic amphibian, one 750-hp. Hornet

Selecting a Plane

Before an engineer can safely commit his client to the purchase of any type of plane, a study of climatic conditions is desirable. U. S. Bureau of Mines *Information Circular* 6767, by Hugh M. Wolfin, contains interesting data, which cannot be reviewed here for lack of space. Thus, planes of more than one motor are unacceptable in cold climates, because with limited facilities it is impossible to heat more than one motor at a time with an oil starter. In the same way, a plane landing at 75 miles per hour would not be suitable for short landing fields and small lakes. All metal planes are relatively more serviceable in arid countries than wooden spar types. Planes that can be hauled out on the beach are more secure in case of storms than ones left at moorings. The newer types of amphibians are capable of landing in rough water, whereas seaplanes require comparatively smooth landings and take-offs.

Conclusions

Depending upon the aggressive type of management of a mining or oil company in staking and leasing new areas, almost any expenditure for instantaneous transportation may be worth while. Although aviation is finding continuous and expanding application in many operations, it is quite apparent that in some instances, companies indiscriminately purchasing airplanes have become disillusioned.

After the fact-finding period is passed in a company's history, it may become relatively less desirable to own a plane, and more economical to turn to flying services operating in that area.

Aircraft manufacturers have kept pace with the demand for equipment, but the science of aerial geologizing is at present a relatively undeveloped art. According to Woolnough, Geological Adviser to the Australian Government, "it would be false economy to embark upon an extended survey until the necessary preliminary ground work had advanced to a stage when interpretation of the visual and photographic

results was placed upon a firm foundation of ascertained fact." This opinion appears to be shared by many others.

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DISCUSSION

C. H. Beal,* Los Angeles, Calif. (written discussion).—The publication in textbook form of the material suggested in the Committee's Aerial Survey Notes† would be extremely beneficial to the working out of geology, etc., from the air and also in one's office, provided there were contact prints and mosaic maps of areas.

I have been interested in aerial maps and their use in petroleum geology for several years and have done a considerable amount of flying in that period for the purpose of observation from the air, thus gaining ideas of structure, faulting, etc., for transferring to the field parties on the ground for detailed work. The only advantage that current flying has over air maps is that an area flown over at different times of the day or in different seasons of the year shows different things, whereas an air map shows only one view—the view of the camera at the time the picture was taken. I have flown parts of our own San Joaquin Valley at noon, when the sun was at its zenith, and have seen certain things, and I have gone back over the same area when the shadows were long and have been able to detect many other things—mainly alignments of physical features, indicating faults, etc.

Recently, Fairchild Aerial Surveys, Inc., photographed approximately 10,000 acres for me, which included the ranch I own in Ventura County. I have available the detailed geological map made without the use of an air map. I have been able by the use of the contact prints to change materially the structural lines—principally faults—that were on the geological map. I find that the ordinary reading glass under a good light enables me to mark with a sharp pencil the outcrops of many beds and some of these beds may be followed for several thousand feet in this particular area. I was able to get the general dip of various formations and, solely through the use of the aerial maps, and in the office without checking in the field, have been able to locate a water supply of considerable importance for the ranch. Obviously, the office work should be supplemented by field work. The point, however, is that the general structural lines and probable location of faults may be easily detected in the office and the critical points selected for field observations.

One application of air maps not listed in the Committee's form letter is the location of property lines. This was used by the Fairchild Company in the survey mentioned above. Two of the several sides of the ranch had been surveyed, but no monuments existed. The area is very mountainous. Nevertheless, the engineer had made a closed traverse of the boundaries of the ranch. I furnished the Fairchild people with this closed traverse and they were able, without difficulty, to pick out on the air map the exact location of the missing corners. True, this location may not be as exact as would have been determined by a surveyor, but it is quite probable that the location of the corners was within one or two feet, which would be sufficiently close in mountainous country, where surveying is most expensive. If the country were flat and such a minor error were important the actual surveying of the lines would not be so expensive.

The greatest advantage of an air map on this particular job was, I believe, the locations of faults.

I have flown considerably in Alaska and British Columbia, and it appears to me there is an enormous field for aerial observation which will result in time in the location of new mineral deposits.

Would it not be an investment well worth while for the Government to sponsor a project for mapping a great section of the possible mineralized portions of the West; for instance, all of Nevada, or all of Northern California, or all of Idaho? Then to

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* Petroleum Geologist and Engineer.

† See page 598.

turn over to a corps of well trained engineers the study of the contact prints and the mosaics, these engineers to publish the spots that appeared to them to have possibilities for mineral deposits. I am omitting all reference to possible oil fields, though perhaps I should not, for the reason that I have felt for many years that in California the location of geologic structures suitable for the accumulation of oil and gas is controlled primarily by deep-seated faulting. This applies particularly to the prospective area in the flat regions of our interior great valley, where no surface outcrops are visible. I believe a project of this sort would pay for itself a hundred times over in 10 years, and would in fact be similar to the project initiated a year or two ago by the Canadian Government, when it employed a large number of geologists and engineers for the purpose of pointing out the best areas in which prospectors should work.

Certainly a study of air maps and their uses should not be restricted to only a few geologists, who happen to be working for mining or oil companies.

L. Wernecke,* San Francisco, Calif. (written discussion).—I know from experience that most of the items of the Aerial Survey Notes are practical. However, I would like to add that any observer from the air, or of photographs taken from the air, should have had previous surface experience with the terrain being observed, in order to identify and interpret exposures with any degree of professional accuracy. This observer must at all times be fully conscious of the three-dimensional orientation of rock structures and their intersections with irregular surfaces. At high altitudes relief is not as apparent as at the surface and a greater danger exists of misinterpreting outcrops of flat or low-dipping geological structures.

I can add two items to your survey notes: (1) The mineralized aureoles, either oxidized or otherwise, near the margins of granitic stocks are easily recognized from the air, and the more favorable areas for prospecting these margins are readily seen. (2) Mapping in the larger unexplored areas of the North is quite feasible without photography. This is done by using a sensitive compass and measuring distances by timing the flight of the plane between points. Intersecting shots with a small split-sight alidade on a sketching board will roughly locate prominent points either side of the aerial traverse. In a large area (10,000 to 100,000 square miles) it is advisable to land occasionally at some favorable lake or stream to make astronomical observations, using a radio for the exact time from the Arlington Naval Station, and a sextant, or light exploratory transit, for the vertical angles.

W. G. Jewitt,† Trail, B. C. (written discussion).—The Aerial Survey Notes on methods or uses of aerial survey appear complete. In prospecting for metalliferous deposits, notes 5 and 11 cover the most general uses. Some quartz veins and gossanous outcrops also can be recognized from the air.

In the Canadian shield the most common use of the airplane is as transportation to supply prospectors and preliminary development camps. This function has been discussed in a number of publications.

H. M. Butterfield,‡ Toronto, Ont. (written discussion).—The principal use of aviation in mining made by Noranda Mines, Ltd., has been as a means of transportation of men and supplies.

That aviation has not been more widely used for purely geological mapping, or for prospecting, in Quebec and Ontario is largely due to the nature of the terrain. Aerial photography has proved itself valuable as a comparatively inexpensive means of establishing accurate topographic maps that are useful both to the prospector in

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* Mining Engineer and Mine Geologist.

† The Consolidated Mining and Smelting Company of Canada Limited.

‡ Chief Geologist, Noranda Mines, Ltd.

what would otherwise be sketchily mapped areas and to the geologist doing more or less detailed mapping in either new or better known districts.

The direct reading of geology from aerial photographs or observations is attended by considerable difficulty in this part of the pre-Cambrian shield. Major fault zones often find topographic expression and are then discernible in photographs. Physiographic changes are often, of course, clearly shown, but lack of such changes must be interpreted with caution. There is at least one instance in Quebec of discovery of an ore deposit by observation from the air of a gossan zone. Aside from these exceptions, however, the uses mentioned in the Aerial Survey Notes have only limited application in this area, due to the widespread mantle of glacial drift and postglacial lake deposits, and the almost universal heavy tree and undergrowth cover. Differences of timber growth are useful for outlining drift-covered areas, but these comprise at least 80 per cent of the total land surface. Locally, structures may occasionally find expression in tree patterns, or probably more frequently in shadow effects. In the comparatively small areas of burned-over ridges, folded structure sometimes shows up directly. Otherwise unsuspected outcrops in areas where outcrops are scarce may be easily located by aerial photographs, particularly if taken in May or early June.

In general, I think it is safe to say that in Quebec and eastern Ontario a geological interpretation of aerial photographs should not be relied upon without at least a reconnaissance knowledge of the geology of the area under study.

H. S. Robinson,* Schumacher, Ont. (written discussion).—To date considerable areas have been photographed by the Dominion Department of the Interior and these maps and photographs are available to mining companies. The terrain is usually heavily wooded and of low relief. Original photographs that have from 40 to 50 per cent overlap are very useful. The photographs are usually available before maps are made up and distributed and can be used to make base maps of new districts, to determine drainage areas, water supply and power sites. Some geological features are readily discernible with a limited knowledge of the surface geology, such as trend of folding and faults. These are usually indicated by tree patterns. While flat-lying beds are more readily identified by such patterns, they can also indicate steeply dipping formations. No doubt with practice more geological detail could be deduced from the photographs. In a certain section of the Porcupine district the photographs showed claim lines and trails very well and these could be marked out on the photos. The photos were then taken into the field and used in conjunction with a plane-table survey of the ground.

C. O. Stee,† Siscoe, Que. (written discussion).—I have secured from our chief geologist, Dr. O. L. Backman, the following illustrations of change in physiography due to geological structure that might be detected from aerial surveys. These have particular reference to Siscoe Island and vicinity:

1. The deep bay at the northwest corner of Siscoe Island is due to deeper erosion of the softer K-zone talcose schists than the adjacent granodiorite to the north and the volcanics to the south. This is well illustrated on aerial photographs of the island.
2. Blouin and Stabell Lakes are along the Blouin Lake fault, well known at Greene Stabell mine.
3. The chain of islands stretching out to the northwest of Siscoe Island lie along the major structural trend.

* Geologist, McIntyre Porcupine Mines, Limited.
† Mine Manager, Siscoe Gold Mines Limited.

S. H. Haughton,* Pretoria, South Africa (written discussion).—This office is mainly using air-survey photographs of areas of well contrasted topography for the purpose of geological survey. From the air photographs planimetric maps with or without form lines at approximately 100-ft. contour intervals have been constructed on a scale of 1:50,000.

In the field, geological details are inserted directly on enlarged photographs (scale approximately 1:8000) and these details are subsequently transferred to the contour maps.

Our experience has shown that stratification, faulting, igneous dikes, direction of jointing, differences in weathering features between various types of rock can all be readily recognized on the photographs; and, in consequence, the work of geological map making is much more rapid than in areas where photographs are not available. Moreover, in a country such as ours, where topographic maps are practically non-existent, the maps produced from air photographs are more accurate and more detailed than if made otherwise.

B. Lightfoot,† Salisbury, Southern Rhodesia (written discussion).—The following suggestions are offered with reference to the Aerial Survey Notes:

Difference in Timber Growth reveals in addition faults, water and mineral-bearing fissures. Not only timber but all types of vegetation show the geological features; changes can also be noted in grassland.

Stratification and Faulting also appear in bush areas by tree patterns and shadow effects.

Tree Patterns show strike lines in contorted and folded beds.

Transverse Structures.—Dolerite dikes stand out well in granite or gneiss areas. Their detection is often of value in the search for water supplies.

Change in Physiography is also shown by hard and soft bands in pre-Cambrian sediments.

No Rock Outcrops.—The geological value will be greatly improved when color photography is used to trace the changes in soil coloration in deeply weathered areas.

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* Director, Geological Survey, Union of South Africa.

† Director, Geological Survey, Southern Rhodesia.

Geological Interpretation of Aerial Photographs

By J. J. van Nouhuys*

(New York Meeting, February, 1937)

The economics of aerial survey and the technical processes by the aid of which vertical and oblique aerial photographs are turned into line maps showing the most profuse topographical detail such as contours, rivers, roads, plantations, habitations and cultivated lands do not concern us here. They have been ably and exhaustively described in the publications of the British War Office, handbooks on surveying methods and on photogrammetry and in the South African mining, engineering and geological press.

Suffice it to mention that for geological interpretation vertical photographs are used on such scales as 1/5000 to 1/10,000, overlapping each other by 55 to 60 per cent in the direction of flight and some 10 to 20 per cent sideways on to the next sequence or strip. Through this duplication in the overlap the stereoscopic examination becomes possible.

The standards by which monochrome aerial photographs are interpreted can be grouped under the following headings:

1. *Continuity.*—The continuity of topographic relief not obtainable on the ground owing to the smallness of man relative to that relief. Here lies the greatest value of aerial survey with relation to geology. An immediate alignment can often be made of drainages running in different directions on each side of a hill; saddles and gorges in the mountainsides; abrupt termination of outcrop or indubitable signs of faulting.

Most conclusive evidence is thus often obtained about faults which may be perfectly visible at one place on the ground but cannot elsewhere be identified because of decreased throw.

Similarly, dikes can often be followed notwithstanding frequent interruption and disappearance under the soil.

Scattered outcrop in scrub-covered or bush-covered country can be conveniently located and often correlated, with a large saving of time compared with field work.

The same applies to geological contacts. Once these have been established at a few points on the ground, the intervening stretches can be drawn in.

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* Chief Geologist to the Aircraft Operating Co. of Africa (Pty.) Ltd.

2. *Texture.*—Texture is dependent on: (1) actual outcrop and (2) the microtopography of rock surface more or less deeply weathered and covered by soil, gravels and boulders and (3) last, but not least, by the formation of vegetational cover. The mosaic of small areas of cover and of small open spaces give the texture, often most characteristic, for a particular rock (covered not too thickly by soil or detritus).

Suitable terms to designate the highly individual textures of actual outcrop and intervening soil and vegetational cover of the various rocks are hard to find. Leprous, lichenous, shagreen, velvety, scabrous (botanical), reticulate, come to mind but are hardly sufficiently circumscribed, whereas the mechanically reproduced patterns of actual fabrics are too much so.

3. *Trees and Bush.*—Conclusions as to underlying rock formations can often be drawn from the presence or absence of bush and trees and their relative heights and densities, but in no other province are blunders so readily made, as no other feature is so easily affected by direct or indirect human agency. Frequently a good deal of creep takes place. The physical and chemical sphere of rocks may extend and almost invariably does so down the screes, slopes and drainages, and conversely on these drainages the physical and vegetational appearance is changed by leaching and/or usually thicker soil cover.

4. *Tone Value.*—When detail becomes so small that it can be no longer recognized as individual objects it coalesces and forms patches of darker or lighter hue and lesser or greater extent. A further evaluation of visual characteristics now becomes a necessity. Objects in actual vision and on aerial photographs become recognizable by virtue of their shape. The shape in actual vision is defined by the contrast of color and of tone value; i.e., the degree of light intensity in which a color tint appears to the eye, distinct from the color.

While leaving it for the moment an open question whether the sense of tone value of the human eye is an absolute one—that is, not influenced by the color—it is an indisputable fact that the tone-value sense of the eye ánd that expressed by a photograph are entirely different. The latter is the resultant of the light intensity given out or reflected by the object and the photochemical value of the object-color for the particular film or plate used. This falsification of the tone value can to some degree be rectified by the employment of filters and panchromatic films, but these do little more than minimize the predominance of a certain color or wave length of light or increase the range of color sensitivity.

The tone-value discrepancies compared with actual vision remain. That this does not strike us more forcibly in most photographs is because of the fact already referred to—i.e., that we recognize objects by their shape—but the contrast that brings out that shape is not easily memo-

rized. The falsification, therefore, is not a harmful one as long as one remains constantly aware of the subtle changes wrought.

A striking example of the above was furnished by a road traversing a belt of soils derived from igneous schists including several pseudo-horizons of iron ores. This road, perfectly visible from the air (lighter than the lightly bush-clad, grassy country) had practically disappeared on the photographs. When examining the site in the field it was found that the soil was stained intensely red and contained numerous minute black iron-ore fragments. The red, brightly lighted road had the same photographic tone value as the grassland with its innumerable small patches of shadow between the yellow grass. Burnt grass when viewed at a low angle on the ground appears to be very black. Yet on the photograph the same patch may sometimes be shown lighter than the unburnt pasture, because of the actual visible area of the black stubble being very much reduced compared with the light-colored soil when viewed in plan.

Notwithstanding the strong contrast with which the igneous dikes in the dolomite on the western Rand stand out as light ribbons on the photographs, examination *in situ* fails to disclose why this should be so. It is not improbable that quadrant counts should disclose numerical differences in the constituents of the vegetational cover. The differential light absorption may also have something to do with it.

In connection with this last suggestion it must be mentioned that it seems to be supported by the fact that seasonally the vegetation on the dike soil often seems to be advanced compared with the surrounding country. The problem, however, must be even more complex if the possibilities of the richer, deeper soil resulting from the weathered igneous rock and the action of these, at lower levels, impervious barriers across the water table, are taken into account.

Although a really experienced interpreter can get much information from a purely stereoscopic study of the photographs, without field work, it is essential that all aerial geological work should entail a combination of photographic study and field check.

Summary

1. The interpretation of aerial photographs for geological purposes is merely another tool in the geologist's hands. It should normally be applied in conjunction with other methods of approach such as ordinary field work, geological survey and drilling.

2. The photographs frequently reveal geological information not procurable by other methods. In any case, they remove almost entirely the necessity for plane-tabling and even where the geological information

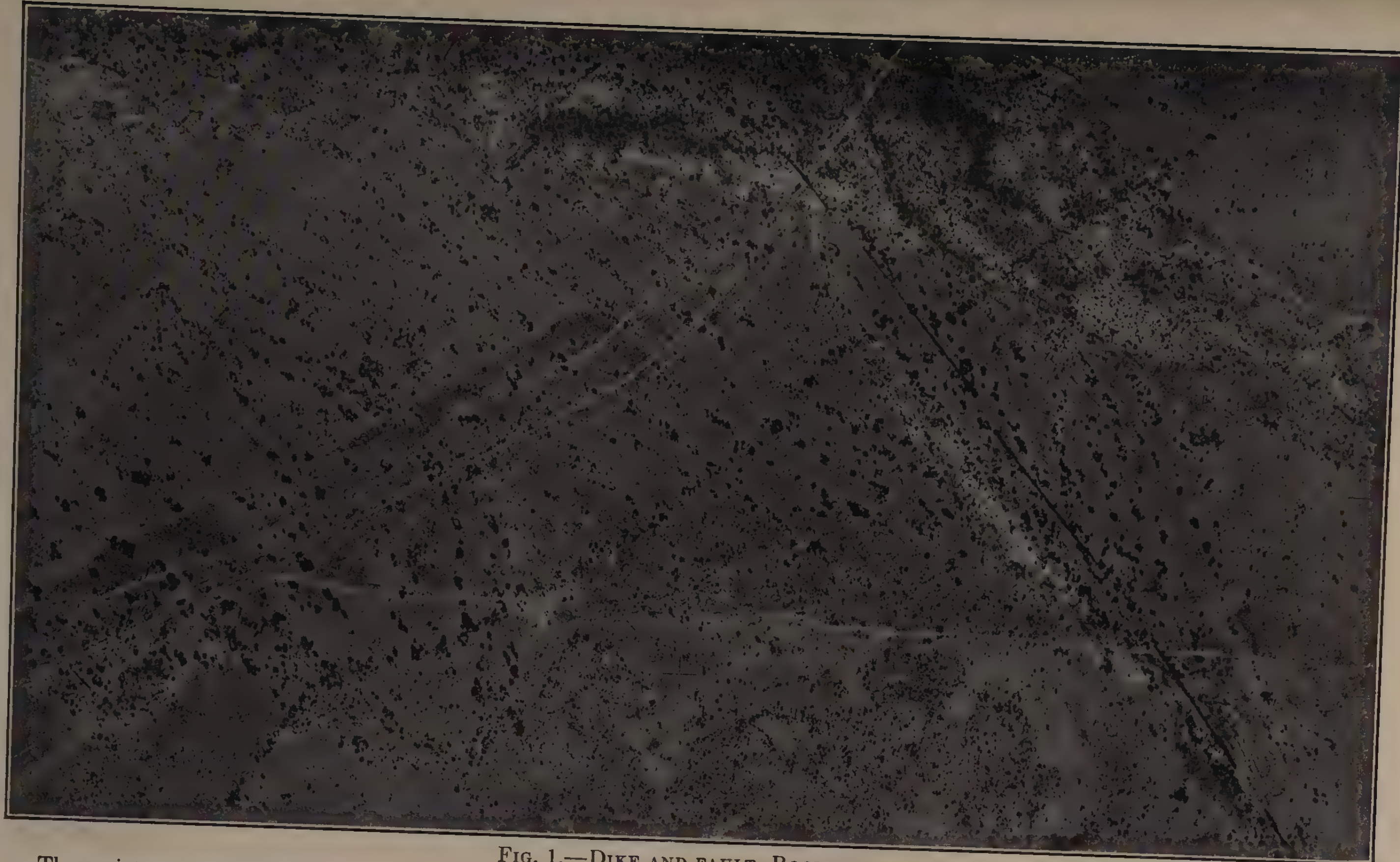

FIG. 1.—DIKE AND FAULT, ROOIBERG.

These pictures are identical, but descriptions have been surprinted on the upper one. The same procedure has been followed on Figs. 2 to 7. Notice Dike and Drainage on Fault in upper photograph.


QUARTZITES
REEF
DOLOMITE
BLACK
DOLOMITE

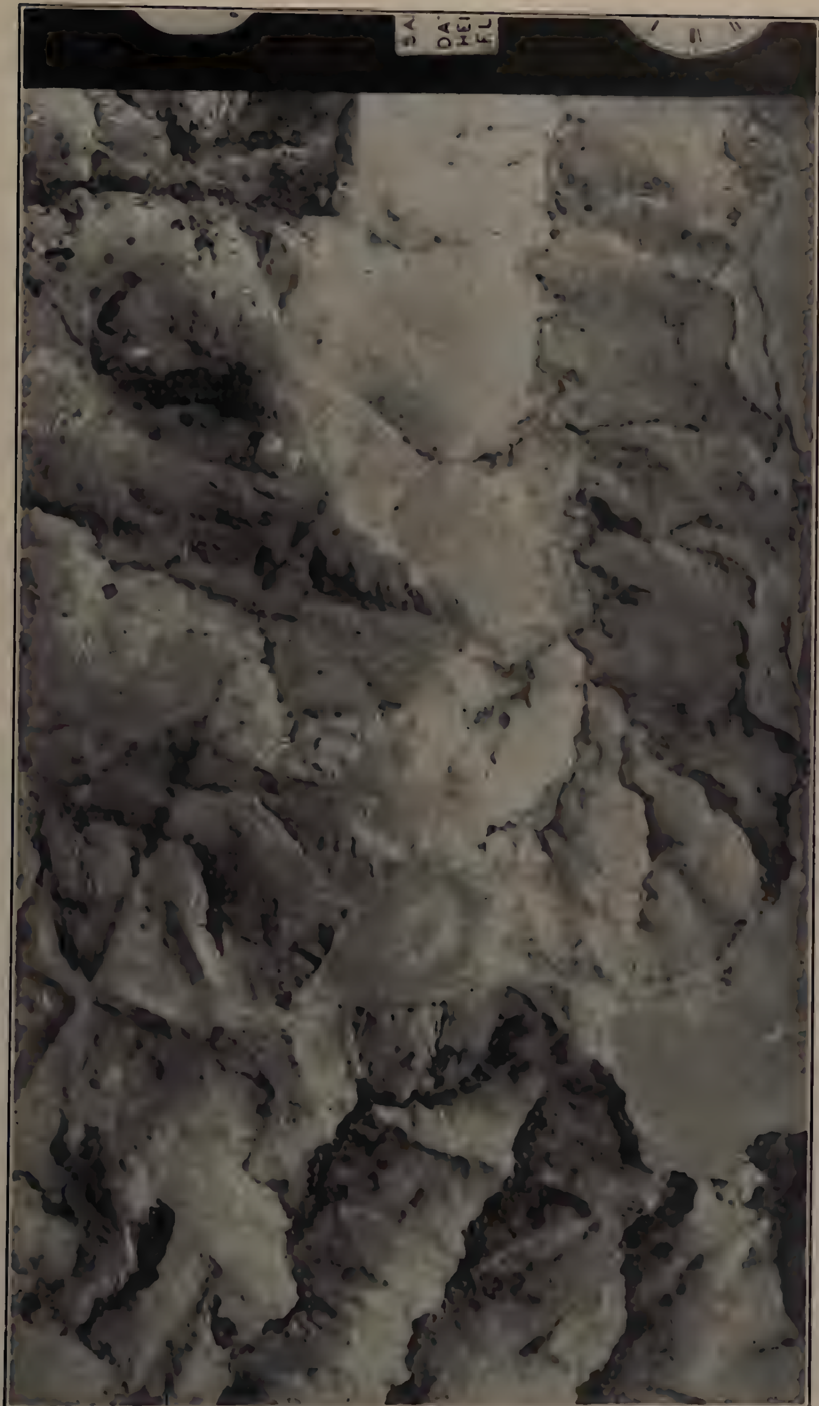

FIG. 2.—DIKES CROSSING IN GRANITE, SABIE-OHRIGSTAD. ESCARPMENT OF BLACK REEF AND DOLOMITE WITH OUTLIER.

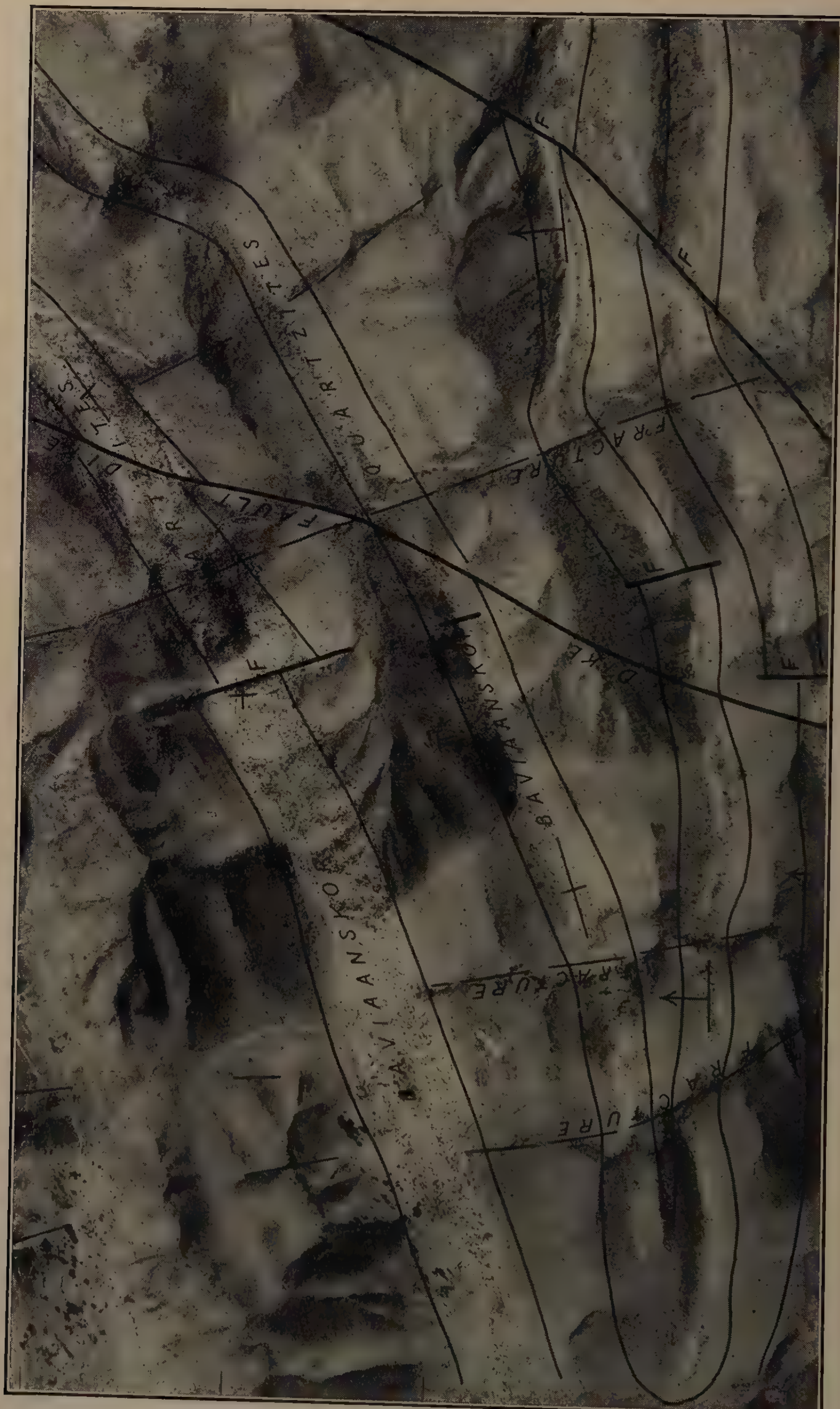
QUARTZITES
FAULT
FRACTURE
DIKE
BAVIAANSKOP
BAVIAANSKOP
FRACTURE

FIG. 3.—PLICATED AND FAULTED BAVIAANSKOP QUARTZITES, BARBERTON.


FRACTURE
FRACTURE
FRACTURE
FRACTURE
CONTACT
CONTACT
GRANITE
JAMESTOWN SERIES
(SWAZILAND SYSTEM)
DIKE
DIKE
FAULT

Fig. 4.—Granite Jamestown series contact, with faults and dikes.


FRACTURE
FRACTURE
FRACTURE
FRACTURE
DIKE
DIKE AND FAULT
DIKE
15°
15°
DIKE
DIKE
Center cross
DIKE
FAULT AND DIKE
FRACTURE
FRACTURE
FRACTURE
FAULT
FAULT
FRACTURE
FAULT
FAULT
FAULT
FAULT
DIKE AND FAULT
FAULT

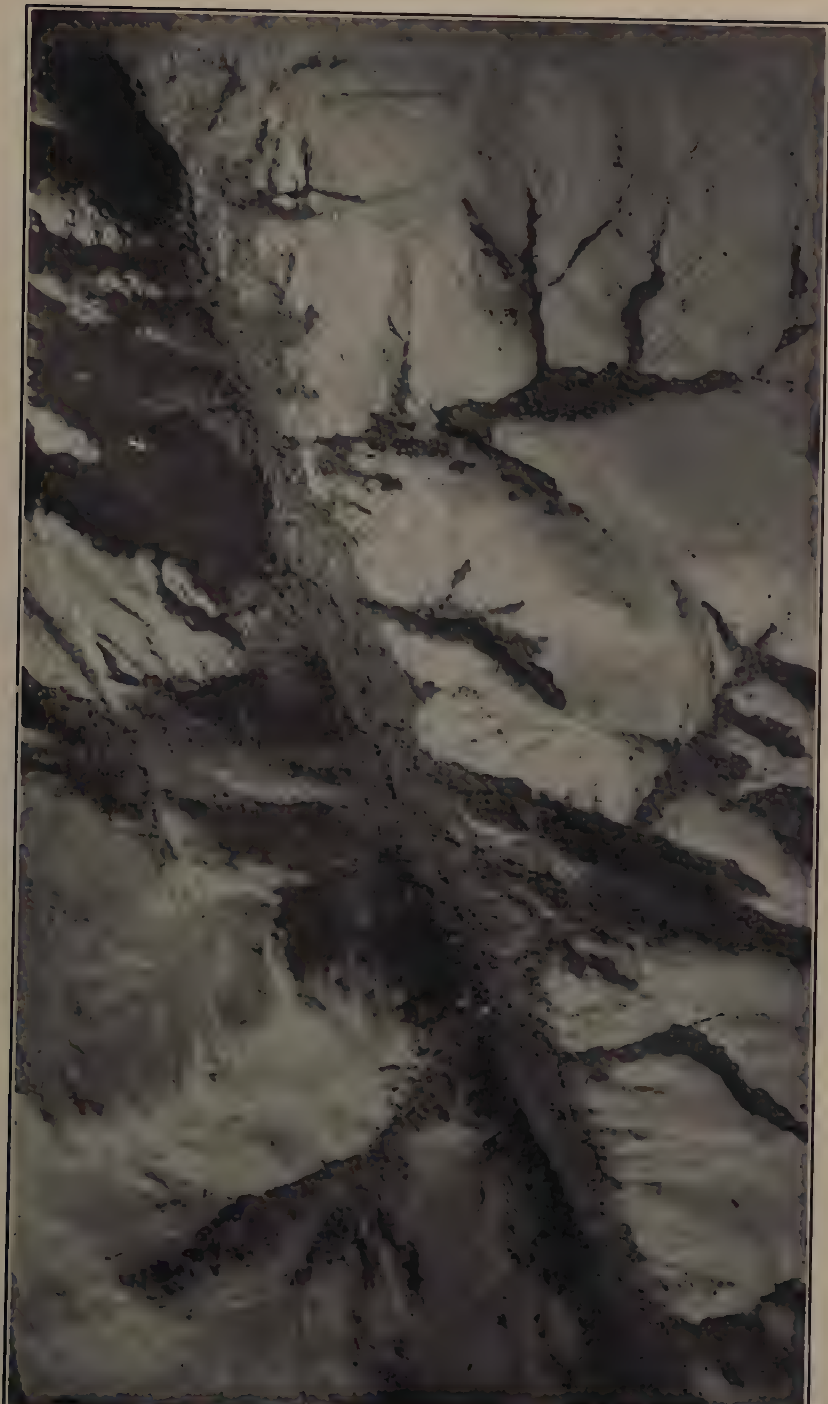

FIG. 5.—WEATHERED OUT DIKES AND FAULTING IN MOODIES QUARTZITES (SWAZILAND SYSTEM), BARBERTON.

FIG. 6.—CONTACTS AND FAULT, NORTHERN RHODESIA VERTICAL.
Notice Quartzite and possible Granite above and Limestones below in upper photograph.


NATIVE HABITATIONS
WITH EUPHORBIA
ENCLOSURE
DIKE
RIVER
BED
OXBOW
RIVER
RIVER

FIG. 7.—OLD AND PRESENT-DAY RIVER BEDS NEAR SHORES OF LAKE VICTORIA. OXBOW AND DIKES(?).

is scant, the photographs are of great value in reducing the cost of the mapping. Further, where the area photographed is of some extent the preliminary examination of the photographs before the field work starts not only permits the geologist to lay out his line of attack but may also allow him to select those areas likely to be of greatest economic interest. These advantages enable work to be concentrated and systematic.

3. The interpreter, apart from a sound basic knowledge of geology, botany and pedology, must also possess the ability to appreciate the interaction of the factors that have produced the appearance before him. He must, in fact, possess a mind with a strong ecological bias and must have used this outlook on a vast number of photographs. It has largely been lack of this mental equipment that has sometimes caused geologists, eminent in their own sphere, to fail to achieve the anticipated, and actually existent, results. Moreover, a preconceived idea of what a particular aerial survey should elucidate is bound to lead to disappointment.

4. No hard and fast rules can be laid down as to the geological meaning of a certain appearance under the headings of Texture, Trees and Bush or Tone Value. Time of year and seasonal changes, general climatic conditions and orientation of slopes, human agency like grazing, burning, woodcutting and cultivation must be carefully taken into consideration. To get the utmost out of a photograph requires a great deal of experience in addition to an acute visual and mental perception.

5. Geological interpretation of air photographs is thus partly art and partly science and can, therefore, at this early stage, be adequately explained only by means of samples. It is to be hoped that as its technique progresses it will not become flooded by the pseudoscientific and obscurantist jargon to which it seems peculiarly liable.

6. There is no doubt that the use of aerial photographs for geological purposes speeds the work and permits it to be more accurate and more detailed. These, in turn, reduce the cost and allow further concentration on areas of greater interest. Also, in considering the cost it must not be forgotten that the photographs provide a permanent record, which can be re-examined at will by others; that topographical maps can be made from them; and that, although this article deals mainly with the geological aspect, much other information (water, soils, vegetation, communications, etc.) is available.

INDEX

(NOTE: In this index the names of authors of papers and discussions and of men referred to are printed in SMALL CAPITALS, and the title of papers in *italics.*)

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